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Card Index.

Commencing with this issue, we print on one of the advertising pages the table of contents of our preceding issue in such a form that it may be cut out and pasted on a card. The contents of our different issues may thus be kept in the convenient form of a card index, until, at the end of the year, the volume index appears.

Manufacture of Brass in the Electric Furnace.

We have pointed out on several occasions the present commercial developments in the manufacture of special steels in the electric furnace. We noted that there are several marked points of superiority in the new method. These were enumerated briefly as follows: First, larger units in place of small units, and consequently smaller labor charge per ton of output; second, durability of the electric furnace, as against the small and costly "graphite-clay" crucible; third, ease of control over purity of product, due to the fact that the heat is generated inside the furnace, and can thus be easily regulated. Just these same factors enter into the manufacture of brass in the present crucible process. There is is an extremely high labor charge, for the brass founder's skill in producing a first-class billet can immensely increase the efficiency of the rolling mill. Naturally, such labor deserves and receives high wages. The crucibles, costing two to three dollars each, only melt from 150 to 250 pounds of metal at a time, and their life is short, not over thirty-five "melts," as the maximum average. Naturally, with the small weights of metal, it is hard to control the quality of brass produced. Any accidental large amount of impurity in the particular slab of spelter or in the particular ingot of copper will manifest itself immediately in the particular billet of brass produced. If, on the other hand, the "melt" had been of two tons size, this accidental impurity of a single piece of the raw material would have been obliterated by the general average purity of the metals. Add to this freedom from dirt, and ability to control heat to 10° C., we see there stand out in bold relief great innate points of advantage on these scores. Moreover, there is also a certain loss in metal in melting due to the volatility of the zinc, and to the breaking of crucibles in the furnace. This would be nearly eliminated in the new process. Other distinct improvements might be adduced, but the above are enough to show what has happened in the case of crucible steel will happen in the case of brass-melting. What is needed is the combination of good practical electrometallurgical experience with good practical brass experience.

The replacement of the old crucible process is sure to come; perhaps it will come in two years, but it will, necessarily, in ten years. Just as the more positive and stronger metal, as iron, throws down the weaker metal, as copper, from solu-

tion, so will the process, stronger in its fundamentals, replace the old and well-tried process that was developed under old conditions. In both cases, however, the question of the time for the completion of the reaction depends on conditions; and in the industrial replacement, this interval cannot be estimated with any accuracy. Nevertheless, it must and will proceed, unless something, not known at the present time, occurs to disturb the conditions. The inevitable advance along this industrial line will have great influence in the metal world, and will also provide in large cities a new kind of load for the central station.

Tantalum and the Tantalum Lamp.

In this issue we publish an article giving the first information on an investigation of the preparation and the properties of pure tantalum, carried out by Dr. von Bolton in the chemical research laboratory of the Siemens & Halske Company in Berlin. These researches have led to the construction of the tantalum lamp, which, if judged from the results obtained by Siemens & Halske, promises to become, in future, of great importance in the electric lighting industry. In our article, on another page of this issue, we cover specially the chemical side of the subject. Electric lighting is generally considered to be far removed from chemistry and electrochemistry; and in fact if a type of lamp is given and the problem is to install lamps within a given district and to supply electrical energy from a central station, no chemical considerations are involved in the solution of the problems. It is very different, however, with the manufacture of electric lamps, and it does not seem exaggerated to state broadly that no important progress in the invention of new electric lamps can be made without extended chemical research. The fundamental idea of the incandescent lamp was simple enough, but it is generally not appreciated, what an enormous amount of chemical research had to be done by Edison in the invention of a suitable filament; yet Edison's successful work in this field places him in the front rank of the practical chemists of his time. All later developments in electric lighting were intimately connected with chemical research; we may simply refer to the use of impregnated carbons in arc lighting, such as in the Bremer lamp, and to the development of the magnetite lamp and of the Cooper Hewitt lamp; all these developments would have been impossible without chemical research; and if investigations along the lines indicated by the last mentioned lamps should lead to an absolutely new and highly efficient method of lighting, the results must be based on chemical experimental work on one side, and on the principles of thermodynamics and physical chemistry in general on the other side.

To come back to the incandescent lamp, it is surely significant that the two most important recent developments in electric incandescent lighting—the Nernst lamp (which its inventor wanted to call "Elektrolytlicht") and the Welsbach osmium lamp—were the direct outcome of chemical research. One principle on which inventors have been working in their research work in this field, was to produce a filament of a material with a very high melting point. The same principle was the leading idea in Dr. von Bolton's research on tantalum. The results of his work are also interesting in various general respects. First, after Auer von Welsbach's renowned work, we have here another instance showing how important the results of researches on rare elements can become; tantalum was so far only a chemical curiosity and of no practical use. The main reason for this appears to be—and this emphasizes another general point—that up to the present, pure tantalum and its properties were unknown. What Moissan prepared in the electric furnace seems to have been a carbide of tantalum. The fact that pure tantalum has new, unexpected and highly useful properties, should encourage similar work with other elements. There is already much being done along such lines. The universal use, in the old metallurgical methods, of carbon as a reducing agent has resulted in metals and alloys containing carbon. Since we have learned to produce metals and alloys free from carbon either by the aluminothermic method or in the electric furnace, their advantages have been recognized. In the same respect Burgess's and Hambuechen's pure electrolytic iron, absolutely free from carbon, is interesting and should prove a most valuable starting material for researches on the properties of steels, for it is a fact that up to the present in all researches on the influence of additions of various metals to steel, the influence of carbon in the iron has never been eliminated. While to the general public the importance of Dr. Von Bolton's work will appear to rest in the new lamp, we may hope that his success will emphasize the importance of chemical research, and will encourage the establishment of research laboratories by our manufacturers.

Recent Developments in Cyanide Practice.

In this issue we publish an interesting paper by Mr. Torrente proposing a new continuous process of settlement and separation of the slimes from the cyanide solutions in the decantation process. The method has much to recommend it and has proven successful on a large experimental scale; it will be interesting to await its behavior on a real commercial scale. In the meanwhile, the paper of Mr. Torrente is being discussed by his fellow-members of the Chemical, Metallurgical and Mining Society of South Africa. Thus, Mr. Dowling points out two points of difference between Mr. Torrente's scheme and present practice on the Rand, namely, first, the use of the spitzkasten principle against its disuse in most modern plants, and, second, the introduction of pulp quietly without agitation; against the tangential delivery now gaining favor. Mr. Dowling concludes that since present practice is the result of years of study and experience, and is giving satisfactory results, it would seem a doubtful policy to return to discarded methods. Against this, one might say that in many cases, methods which were already discarded have later on proven successful under new conditions. Mr. Laschinger points out that a continuous treatment not only requires the establishment of the proper conditions of working, but also the uniform maintenance of these same conditions, and it is the latter problem that is most difficult. The continuous treatment involves the danger that a small amount of foreign matter or slimes with peculiar properties may throw the whole plant out of working at any time. It is the old problem—so important in many metallurgical and chemical processes—of continuous versus intermittent operation. But this general

problem really deserves a more detailed discussion than can be given in this short note.

Another problem of most timely interest in cyanide practice is the proposition—which, we believe, was first made by Mr. Denny—to slime everything. It is, of course, clear that in order to get the best chemical effect, *i. e.*, the best extraction, the material to be treated should be as finely subdivided as possible. As a matter of fact, this has really proven to be correct in practice. Thus, Messrs. Butter and Hamilton, from their experience at El Oro, Mexico, conclude that the gradual increase of extraction with increasingly fine grinding, points to the theory that if the whole could be reduced to an absolutely amorphous and impalpable powder, there would be nothing to hinder the total extraction of the value. Mr. Denny's scheme has, however, other advantages; thus, since there is no longer an independent treatment of concentrate, sand and slime, we have a consolidation of plant, resulting in saving in the first cost of installation. On the other hand, it remains to be seen whether on large scale working these advantages are not counter-balanced by an increased working cost, though the experience of those who used the new method speaks against this and distinctly favors the new method. In view of this industrial development it is certainly amusing to remember that not so very long ago the endeavor was to avoid as much as possible the production of slimes—for the simple reason that one did not know how to handle them.

Consolidation of Metallurgical Experience.

In a recent paper presented by Messrs. Charles Butters and E. M. Hamilton, before the British Institution of Mining and Metallurgy, on regrinding of sands in cyanide practice at El Oro, Mexico, the authors remark that in the earlier days of gold mining, such figures as they give in their paper would never have been given to the public. "But the old policy of secrecy in such matters is gradually passing away, and mining companies all over the world are beginning to see the advantage of thus comparing notes with one another—a course which has resulted in an enormous increase of knowledge in general." It is sincerely to be hoped that this new policy, based on a sense of enlightened self interest, will extend further and further, especially in these days of new enterprises. For it is not at all an exaggeration to say that at the present time there is a more active development in the mining and metallurgical world than there has been at any former period. This new work cannot fail to produce results in the course of time. But as a necessary concomitant to the success, there may be many times as many failures. Industrial progress is like progress in any line—a process of evolution with the destruction of the unfit. Some critic of Darwin once said in a witty criticism on the latter's doctrine, that he did not believe that nature was as foolish as a man who, on a rabbit hunt, should discharge a million random shots in a field and perhaps bag one at this great expense. Nevertheless, nature does make progress in a most expensive and destructive manner.

It is somewhat similar with the development of a new process or a new mine. A certain lot of hard work must be done and a certain amount of good money must be spent to prove with absolute definitiveness that this reaction will work on a large scale, that this construction contains weak points, or that this vein extends beyond the vein which interrupts it. Fortunately, the action of man's intelligence differentiates industrial progress from organic progress. The mind steering the course of the new craft can exercise discrimination. It can reason from analogy and from previous experience that such a reaction will not work on a large scale, because it is impossible to hold conditions uniform and right in the large scale apparatus. Nevertheless, new work, however well planned, has been always expensive and always will be expensive, for the reason that in certain cases human prognostications must be proven before any forward step can be conservatively attempted. Then, too, the expense is heaviest in proving that certain things cannot be done. For it is foolish for any one to say absolutely that anything is impossible. In every case, the probabilities must be balanced against each other. The estimation of these chances is the basis of action. The results should on the average be worth far more than the expenditure.

Such deliberations should act as an incentive to every investigator and developer of new things to give whatever he can afford to the common fund of knowledge. We know, of course, that there are certain points which even the most generous man should rightly keep to himself. Every chemical and metallurgical failure injures indirectly each metallurgist and each chemist, for it justly makes capital more suspicious and cautious about entering on a new enterprise. The optimistic boastings in the columns of the Sunday newspaper about certain alleged triumphs of engineering do incalculable harm. If there are enough well trained engineers, the charlatan will be less prominent. There is work enough to keep all good men at work busy and active. Some narrow spirits do not realize this. We hope that the selfish engineer will not only not receive the confidences of his associates, but that he will also lose directly from the general distrust, caused by the short-sighted policy which he and his ilk engender. The broadest and grandest successes are not envied, but admired, when a broad man is generous in giving information about his trials. In the narrower field of electrochemistry and electrometallurgy the American Electrochemical Society—and, we modestly hope, this journal—have had in the past an influence in systematizing the experience of all and thus making experimental work more efficient. This has reacted on all, and, we trust, has made the obscurantists see the error of their ways. Many of the failures of the past might have been avoided if experimenters had pooled their experience. This fact seems now to become clearly understood, and in the future we can hope that the evolution of the metallurgical industry will not be fraught with too many failures; that the engineers of the future works will not shoot too blindly in the field of practical endeavor; that when a place is to be aimed at they will bring out a gatling gun of experience and thoroughly cover that one spot. To effect this we will try to do our share, so far as our ability and the good will of our contributors extends, as a clearing-house of metallurgical ideas.

American Electrochemical Society.

The date for the spring meeting to be held in Boston, Mass., has now been definitely fixed; it will be held on the 25th, 26th and 27th of April. The local committee, with Prof. W. H. Walker as chairman, is making arrangements so as to make this meeting a most memorable one. There will be especially fine facilities for making exhibits and experiments in the convention hall.

New York Section of the American Electrochemical Society.

The first meeting of this section in the present season was held on January 24, in the hall of the Chemists' Club, Dr. C. A. Doremus being in the chair. Dr. E. F. Roeber presented a paper giving a summary of the various uses to which electricity may be put in iron and steel metallurgy. After some introductory remarks on the use of blast furnace gases for driving gas engines—which has led to a new commercial source of electric power—and on the special reasons which have caused European metallurgists to become fascinated with applications of the electric furnace in iron and steel industries, he touched briefly on the use of electricity for all the various power purposes in modern steel plants, and on the use of electricity for electromagnetic and electrostatic separation—two kinds of concentration of ores—and then passed over to the use of electrolysis for refining iron. The author described the method of Burgess and Hambuechen for producing almost absolutely pure iron by electrolysis, and then referred more briefly to Edison's manufacture of the steel containers of his storage battery by electrolytic deposition upon a mould. The main part of the paper dealt with the discussion of applications of the electric furnace, the following four problems being separately discussed: Production of ferro-alloys; manufacture of high-grade special steels to compete with our present crucible steels; the production of structural steels to compete with Bessemer and open-hearth steel; the reduction of iron ore to produce pig iron. The author emphasized the importance of the experimental evidence contained in the recent report of the Canadian Commission and summarized their conclusions and the evidence on which the conclusions are based. Dr. Eugene Haanel, the head of the Commission, had courteously placed a number of copies of the report at the disposal of the Section.

In the discussion which followed, Mr. J. T. Morehead gave very interesting details on the work now being done in the production of ferro-alloys by the Wilson Aluminium Company. He stated especially that the method of von Kuegelgen and Seward (our Vol. II., page 468) is now used by the company on a commercial scale; the electric furnace process is so conducted that the consumption of the carbon electrodes is a minimum. Under such conditions the alloy will contain much carbon, which is afterwards removed by means of metallic calcium. He also mentioned the use of silicon as a refining agent. The company makes the calcium metal by an electric method. Dr. George F. Kunz gave some interesting notes on natural and artificial diamonds and carborundum with reference to experiments of Moissan; Mr. C. O. Mailloux gave some humorous reminiscences of the time when aluminium was first made commercially, while the difficulty was to find uses for it. Mr. A. J. Rossi, the distinguished titanium expert, had sent a communicated discussion, which was essentially the same as his letter on page 53 of this issue.

Dr. E. Durant then read a highly theoretical paper, in which he referred to an electrical gyroscope which was exhibited and shown in operation.

Tantalum and the Tantalum Lamp.

The attempts of getting a more economical incandescent lamp have mainly gone towards finding a material which has a very high melting point. The Nernst lamp and the Osmium lamp represent examples of work done in this direction. In the chemical research laboratory of the Siemens & Halske Co., of Berlin, Dr. W. von Bolton has been making extended experiments with various materials, and has specially studied vanadium, niobium, and tantalum. The outcome of this research is the tantalum lamp, which appears very promising. Dr. von Bolton reduced potassium tantalo-fluoride in the manner described by Berzelius and Rose, and found that the finely divided tantalum so produced became fairly coherent on rolling, so that by this treatment metallic strips could be made of it. On the other hand, it was tried to bring tantalum oxide into the shape of a filament by mixing it with paraffine, and to reduce it directly into the form of a metallic thread. In these experiments there was for the first time observed a minute globule of molten tantalum, which was of sufficient toughness to permit hammering and drawing into wire. Following up this observation, tantalum powder was melted in a vacuum and then it was found that the highly heated metal parted with the gases it contained. In this manner the first filaments of pure tantalum metal were produced, which, however, were quite small. When they had been used in lamps with promise of good results, an attempt was made to devise a definite process of purification.

Potassium tantal-fluoride reduced to metallic powder, contains a small proportion of oxide and of hydrogen absorbed during the reduction. When the powder was melted in a vacuum the oxide and absorbed gas disappeared and a reguline metal remained, and on carefully remelting, no appreciable impurities could be detected in it.

The chemical properties of this pure tantalum are very remarkable. When cold, the material resists chemical agents strongly; it is not attacked by boiling hydrochloric acid, aqua regia, nitric acid or sulphuric acid, and it is also indifferent to alkaline solutions; it is attacked solely by hydrofluoric acid. Heated in the air, it assumes a yellow tint at about 400° C., like steel, and also like steel the tint changes to dark blue when the tantalum is exposed for some time to 500° C., or for a shorter time to 600° C. Thin wires of it burn, when ignited, with low intensity and without any noticeable flame. It greedily absorbs hydrogen as well as nitrogen, even at a low red heat, forming with them combinations of a metallic appearance, but rather brittle. It combines with carbon very easily, forming several carbides, which, as far as they are at present known, are all of metallic appearance, but very hard and brittle. The product which Moissan thought was tantalum was, according to von Bolton, clearly a carbide of this kind, or an alloy of a carbide with pure tantalum. While Moissan ascribes to the metal the specific gravity 12.8, great hardness and brittleness; these are, according to Dr. von Bolton, not properties of pure tantalum. The specific gravity of Bolton's tantalum, when purified by fusion and drawn into wire, is 16.8. It is somewhat darker than platinum and has a hardness about that of mild steel, but shows greater tensile strength. It is malleable, although the effect of hammering is relatively small, so that the operation must be rather long and severe to extend the metal into a sheet. It can be rolled as well as drawn into very fine wire. Its tensile strength as a wire is remarkably high, and amounts to 95 kilos per square millimeter, while the corresponding figure for good steel is 70 to 80 kilos., according to Kohlrausch.

The electrical resistance of the material at indoor temperature is 0.165 ohms for a length of one meter and a section of 1 square millimeter (specific conductivity as compared with mercury 6.06); the temperature coefficient is positive and has a value of 0.30 between 0° and 100° C.; at the temperature assumed by the incandescent filament in the lamp under a load

of 1.5 watts per c. p., the resistance rises to 0.830 ohms for a length of 1 meter and a section of 1 square millimeter. The coefficient of linear thermal expansion between 0° and 60° C. is 0.0000079, according to experiments made by the Imperial Normal-Aichungs Commission. Fusion is produced by a gradual softening which appears to extend over a range of temperature of several hundred degrees. The specific heat is 0.0365, so that the atomic heat is 6.64, which is in accord with the law established by Dulong and Petit.

The lamp department of the Siemens & Halske Co. appears to have successfully overcome the difficulty of making a tantalum filament for use of the lamp in any position. The energy consumption is between 1.5 and 2.5 watts per c. p.

Current Notes.

Franklin Institute.—At a recent meeting of the Franklin Institute, Prof. Wilder D. Bancroft, of Cornell University, presented a paper on the chemistry of electroplating.

Electrolytic Iron.—We have received from Prof. Charles F. Burgess, a small but magnificent sample of his pure electrolytic iron, the production of which was described in great detail in our Vol. II., page 183. We understand that the Carnegie Institute has granted funds to Prof. Burgess to enable him to continue his splendid researches in this field.

Aluminothermics.—The Goldschmidt Thermit Company has presented to the National Museum at Washington their entire exhibit at the St. Louis World's Fair, at the suggestion of the museum authorities. The collection gives a complete picture of various applications of aluminothermics, and consists of large and bulky pieces of pure, rare metals, free from carbon, such as chromium, manganese, ferro-titanium, etc.; diagrams and models showing various applications of thermit; numerous pieces—in particular, welded girder rails—showing the successful operations performed by the process in joining and fusing iron and steel, and metals and specimens of the methods used.

Electric Furnace Curiosities.—We have received from Messrs. FitzGerald & Bennie, of Niagara Falls, two interesting samples which came out of recent experimental electric furnace runs, and which show clearly the effects of high temperatures from small radiating surfaces. One of them is a section of a long tube of pure quartz, and is pretty well fused on the inside; the heat was applied from a bare carbon rod which had been heated to a very high temperature. (This is precisely the method of producing fused carbon articles, quite recently patented by Prof. Elihu Thomson). The other specimen is from an experiment with a very small carbon rod, surrounded by a porcelain tube; enough heat was generated in this small rod to fuse the porcelain so that it ran down into the lower portion of the tube and finally filled the tube up to the carbon. The specimen shows some light greenish material which is probably some silico-carbide, while some small globules sticking to the small carbon rod are probably metallic silicon, reduced from the silica in the tube in contact with carbon, or they may be aluminium silicide.

Calendar.—The Laclede Fire Brick Manufacturing Company, of St. Louis, Mo., issued a diabolically striking and amusing calendar, on which "His Satanic Majesty" expresses his satisfaction with fire-bricks, which are "the only material to stand my heats."

CORRESPONDENCE.

The Electric Furnace in the Iron and Steel Industry.

To the Editor of Electrochemical and Metallurgical Industry:

Sir.—The subject of the use of the electric furnace in the iron and steel industry is of such a vast importance, and one which has given rise to so many conflicting opinions, pro and con, that it is not possible to do justice to it in a short note. The able report of the Canadian Commission, covering, as it does, the most improved methods used on the Continent, speaks for itself and its conclusions, as a whole, cannot be contested. We will confine ourselves in our remarks strictly to the metallurgical part of the question, leaving to others to discuss the disposition of the furnace, and other details. We may say that, in a general way, the whole question is one of economy of the electric treatment in comparison with other metallurgical processes employed for the same purposes. As to the possibilities, they have been demonstrated beyond question, both as to results obtained and qualities of the products.

Economy is, to a certain extent, a relative word. As regards steel, for instance, crucible steel commands a much higher price than open-hearth steel, which itself is far superior to converter steel; so that the economy of an electric treatment may well be demonstrated for a special quality of steel, while it may be different for another. In what is called the crucible steel method, the metal is melted in a vessel of small dimensions, and is protected against the obnoxious influences of certain gases, such as those of the open-hearth. Electricity can certainly compete favorably with the present method for crucible steel.

Even as regards open-hearth steel the operation involved in the method of making steel can be secured electrically, as surely and as advantageously and under conditions which secure a metal of superior quality, hence commanding a higher price, and thus upsetting the lack of economy were it proved to exist; we mean for special steels. Many people, when electric steel is mentioned, understand it to mean steel obtained directly from iron ores electrically, while all the processes which have been successfully carried on in Europe are what is called the pig and scrap process, or the pig and ore scrap or mixed process, as they are carried out now in the open hearth. In the best apparatus examined and discussed, the electricity is used as a source of heat for raising the temperature of the charge. This electric heating can be done very successfully, very cleanly and even economically when the cost of fusion is much less important than the soundness, toughness and, in general, the superior qualities of the steel, as is the case in the manufacture of the finer grades or special steels.

There is quite a difference between the amount of heat necessary to bring pig iron and scrap to a certain temperature, and to melt them, on one hand, and reduce iron from its oxide to the metallic state, on the other hand; 1796 calories are required to reduce 1 kg. of iron from ferric oxide, and some 200 calories more for impregnating the metal with carbon, phosphorus and sulphur—for pig iron, and also to a less but still important extent for steel—this, with the heating of the materials to the proper tempertature and melting the metal, represents some 2500 calories or thereabouts per kilogram obtained. On the other hand, some 500 or 600 calories at most (say ¼ to 1/5 of the former figure) are sufficient, if no reduction of oxide of iron is resorted to (pig iron and scrap process), or if but a limited amount of it is to be reduced (mixed process).

We do not take into consideration in giving these figures, such heat or part of it as may be supplied by the combustion to CO_2 of the CO resulting from the reduction of ferric oxide by carbon. If we take in consideration the equivalence of heat and mechanical energy, a certain amount of heat corresponds to a certain amount of horse power in a given time, and it is immaterial if the heat appears as fuel burnt (like in the blast furnace), as gases (like in the open-hearth), or as electrical energy developed. It can be easily shown that starting from pure oxide of iron reduced by carbon, with all other contingencies for making pig iron, it is necessary to supply the heat equivalent to some 145 hp. per gross ton of pig iron made in twenty-four hours, and if we adopt an 80 per cent duty for electric smelting, we come to 175 hp. per gross ton-

day. It is true that some of the heat of CO forming CO_2 can be made available. But if, on the other hand, we consider that we will have practically some slag to meet; that there will be a loss of heat by radiation, and a loss of electricity from the supply to the furnace; that the composition of the ores will vary; that even their physical or chemical state will be different, the above figure ought to be considered in regular practice as a very favorable one, due weight being given to causes of occasional imperfect working inherent in this as in any other metallurgical method; in certain cases 200 hp. per gross ton-day would not be exaggerated.

From the "Canada Report" we find that an average of the horse-power-year per ton of steel was for Kjellin's and Héroult's process (mixed process) 0.16 hp., which corresponds to 58 hp. per ton of ingot-day, and for the pig and scrap process (Keller) 46 hp. per ton of ingot-day. For pig iron the figures are: 0.525, 0.53 per hp.-year (Héroult process and first run of Keller); this gives say, 0.53 hp.-year per ton of pig iron or 189 hp. per gross ton of pig iron in twenty-four hours, which agrees well with the figures we have given above.

At this rate, counting even only on the best possible condition, theoretical ones, so to speak, in one case we need between 150 to 200 hp. for 1 ton (gross ton) pig iron per day; that is, for a production of 500 tons a day, such as is frequent and current practice with the Pittsburg blast furnace, 75,000 to 100,000 electric horse-power would be required. We have never claimed that electric smelting could compete for such daily productions with the blast furnace, but we believe that for limited or moderate local production an electric plant of moderate size involving as expenses of installation only turbines and dynamos, and the collecting of smaller powers to a central plant, could be established so as to supply the current at a cost of $8 to $10 per hp.-year or even less.

Under such conditions, with ores near the furnace and which could be had at practically the cost of moving, or say at 0.75 to $1.00 a ton or so, a local industry to supply local wants could be successfully carried on. In electric smelting the carbon being used for reduction only, not more than ¼ to 1/3 of the amount required in a blast furnace would be required; furthermore, its physical state is not like in the blast furnace, a condition *sine qua non* for use. Charcoal, made from twigs and small branches can be used in an electric furnace. The advantage of the latter is the cheapness of its construction, compared with that of a blast furnace. No machinery for blowing nor for the heating of the blast are necessary, no expensive stack is required. It adapts itself to an increase or decrease of production by the addition of a furnace put up at a very moderate cost, or the shutting off of a switch. Finally the metal thus obtained is more homogeneous, close-grained, in short, of a superior quality, and may command and has commanded a higher price.

In short, so far as pig iron is concerned, electric smelting is a question of cheapness of establishment or supply of horse-power; cheapness of ores used in loco, favorable location, limited production for local consumption, or under conditions where the lack of proper fuel excludes the possibilities of a blast furnace plant.

As far as ordinary steel is concerned, the same may be said. Electrical plants capable of making 1800 tons of steel and turning them into rails, as it is done with two converters in twenty-four hours, cannot be thought of, and such converter plants are in daily operation in Pittsburg. As to open-hearth steel, there are now in operation open-hearth furnaces with an output of 200 tons of steel, while 50 to 100 ton open-hearth furnaces are of frequent use, and as far as quantity of steel made in a day is concerned, electric smelting would hardly prove economical. However, from the figures given above, since only some 50 to 60 hp. are required per ton of open hearth steel per day, a plant of 200 tons would only require 10,000 to 12,000 hp. Large as it is, this figure is admissible. If, on the other hand, special steels are made, then the daily production falls considerably below the large figures mentioned. The electric process then assumes an equality as to cost, and certainly a superiority as to quality of product.

For the production of crucible steels, the finer qualities of steel and special steels, electric smelting, we are confident, will at some near future assume its superiority, and will come more and more into use, and the same may be said of ordinary open-hearth steel if moderate productions are admissible.

We should add that in the above we have not intended to give the items of the balance of heat of each process, but merely approximate figures, which are at least sufficient for comparisons.

Auguste J. Rossi.

New York City.

Zinc Made in the Electric Furnace.

To The Editor of Electrochemical and Metallurgical Industry:

Sir.—With reference to the article of Dr. F. Meyer on the present status of the zinc industry, and an editorial discussion of the same subject, published in your January issue, I would like to make a few remarks on the properties and uses of zinc produced in the electric furnace of De Laval.

This zinc was placed on the market in Europe about a year ago. With respect to qualify, this pure zinc G. D. L. is able to compete with the best kind of zinc on the market, such as "Sterling" and "Vieille Montagne extra pur A," as will be seen from the following analyses made by me. Like Sterling zinc and Vieille Montagne extra pur A, the pure zinc G. D. L. is sold under guarantee not to contain more than 0.1 per cent total impurities:

Number of Analyses.	Sterling.	Vieille Montagne Extra pur A.	Pure Zinc G. D. L.	
	9	8	2	
	per cent.	per cent.	per cent	per cent.
Pb	0.03 to 0.05	0.05 to 0.07	0.06	0.03
Cd	.....	....	None	None
As	.....	.....	None	None
Fe	0.01 to 0.03	.....	0.01	0.01
S	.....	.....	0.00 (3)	....

Such pure kinds of zinc are used in the brass industry. For ordinary kinds of brass, "Hohenlohe" zinc (containing in the average 1.1 per cent Pb) or similar kinds of zinc are sufficient, while for brass rods and for certain types of brass wires often some lead is added, in order to facilitate the working on the lathe. On the other hand, it is necessary to use the purer and more expensive brands of zinc for certain purposes, like the manufacture of brass sheet for cartridge shells.

According to the specifications of most European governments, this brass must not contain more than 0.2 per cent Pb and must satisfy certain mechanical tests. It is easy to calculate that for this purpose it is necesary to use the pure brands. Moreover, very small quantities of As and S have, in this case, an appreciable bad effect. If for the manufacture of cartridge shells pure G. D. L. is used, instead of "Sterling" or "Vieille Montagne extra pur A," which have proven suitable in many years' tests, no difficulties whatever are experienced in the method of manufacture, nor does the proportion of faulty products pass beyond the usual percentage.

At present, pure zinc G. D. L. is a little (though very little) cheaper in Europe than Sterling zinc and Vieille Montagne extra pur A.

C. Offerhaus.

Holcomb Rock, Va.

Resistance Furnace for Crucibles.

By FRANCIS A. J. FITZGERALD.

In making experiments with small electric furnaces, it is by no means easy to form an opinion as to what may be done with a very much larger furnace working on a commercial scale. In certain cases it is impossible to predict with certainty what the results will be when working with the larger furnace, and the only method of arriving at a conclusion of any value is to install the large furnace and try the experiments with it. The great difficulty we are met with in the design of electric furnaces is the lack of data on which to base our calculation, for though there can be no doubt that many of these data are in existence, they cannot be obtained. Most processes in which electric furnaces are used are patented, and the details connected with the subject are carefully preserved secrets. Nevertheless, a great deal of experience may be gained from the study of small furnaces that will prove valuable in the design of large ones, the more so in that certain causes of inefficiency due to poor design are made much more obvious when working on a small scale.

One of the most important considerations in the building of all electric furnaces is that of heat insulation, and in no way is this made more evident than in experimenting with small furnaces, for example, furnaces using from 20 to 50 kilowatts. If two blocks of iron of similar form, but of different dimensions, are heated to the same temperature, then, *ceteris paribus;* the rate of cooling will be proportional to the squares of the dimensions, the quantity of heat in the blocks will be proportional to the cubes of the dimensions, therefore the small block will cool faster than the large block.

If a given mass of iron in the form of a sphere is heated to a certain temperature, and a long rod of iron of the same mass is heated to the same temperature, then, under similar conditions the rod will cool faster than the sphere. If two spheres of iron of the same mass are heated to the same temperature, and one of them is surrounded with a good heat insulator, while the other is exposed to the air, the latter will cool more rapidly.

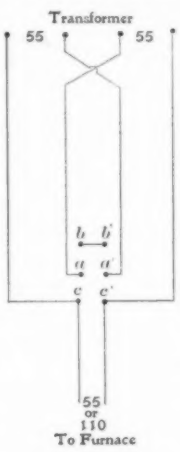


FIG. 1.—DOUBLE-THROW SWITCH.

These observations are absurdly trite, yet they seem to be continually disregarded in the design of electric furnaces. Now, even under the most favorable circumstances, as to price of power, if we simply consider the cost of a unit of heat, that generated electrically is more costly than that generated by burning fuel. Therefore, in dealing with electric furnaces, one of the most important points to be kept in mind is efficient heat insulation. One of the reasons for using electrically generated heat is because with it very much higher temperatures can be attained than with burning fuel. Now, the rate at which heat is conducted or radiated from a body is a function of the difference of the temperatures of the body and its surroundings, and hence heat is conducted away more rapidly in the case of electric furnaces.

This is merely one example of many points that can be studied in dealing with small furnaces, and is a very obvious example, too, as anyone experimenting with a small furnace will soon find out.

Many experiments and investigations can be conducted with furnaces using 20 kilowatts or less. If an alternating current, which is more satisfactory, is used, it will probably be brought to the furnace room at high voltage, and if so, one of those transformers should be used which can be connected up so as to give two secondary voltages. If possible, a transformer and regulating device should be used, so that a considerable range of current and electromotive force can be obtained. This, of course, adds appreciably to the expense of the apparatus, and if only an ordinary transformer is available, giving two voltages, then it will be convenient to have it connected so that 110 or 55 volts can be obtained. For a 20-kilowatt transformer the leads should be connected to a double-throw, two-pole, 500-ampere switch, as shown in Figs. 1 and 2. The upper terminals of the switch b, b' are connected by a copper bar, while the leads from the two 55-volt windings of the transformer are connected to c and a', and to a and c', respectively, so that when the switch is thrown up, connecting a and a' with b and b', the windings are in series, and thrown down to c and c', they are in parallel.

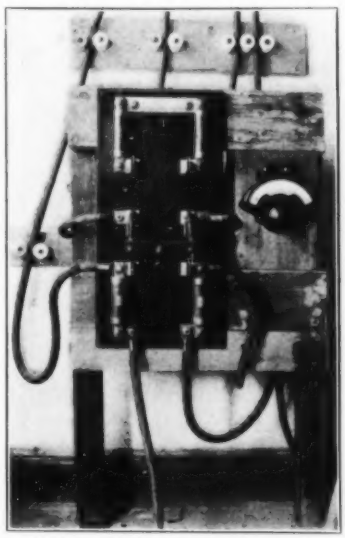


FIG. 2.—DOUBLE-THROW SWITCH.

In this way it is possible to have a maximum current of 180 amperes at 110 volts, or 360 amperes at 55 volts. This is about the cheapest arrangement that can be used, and is also satisfactory for experiments with small arc furnaces. It is, however, very far from being satisfactory for certain resistance furnace experiments where, as we shall see, a current of low voltage and very high amperage is required.

In experimenting, even with so small a furnace as one using 20 kilowatts, it is well to have a good-sized bench for the work. This may be constructed of two-inch boards, supported on 4 x 4 inch legs placed at frequent intervals and properly braced to give rigidity to the structure, which will be called upon to stand heavy weights. The bench, which may be about 10 x 4 feet, having been set up, strips of wood are nailed along the sides and ends, so as to form a tray about 1 inch deep, and this is filled to a depth of about ½ inch with sand.

A floor of brick is now laid on the bench, and on this floor is built the brick work, cellular structure shown in Fig. 3 in isometric projection. The cells of the lower row are empty, but those of the upper row are filled with a good heat insulator, such as asbestos, or, better still, infusorial earth, which is one of the best heat insulators that can be obtained, and if pure, is much more refractory then asbestos.[1]

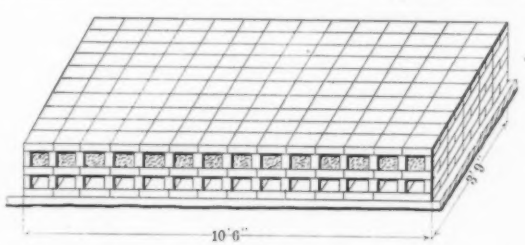


FIG. 3.—CELLULAR STRUCTURE OF BRICKWORK FOR ELECTRIC FURNACE.

[1] I am told that a good grade of infusorial earth is hard to obtain. An excellent material is put on the market by the Lake Region Mining Company.

At first sight this bench may seem to be a needlessly elaborate structure, and doubtless it would be if furnace experiments only lasted half an hour or an hour, but if a furnace run lasts nine or ten hours, a less elaborate structure would not be suitable. Without the heat insulation in the upper row of cells the losses would be very serious, and by having the lower row of empty cells, it is possible to detect accidents, such as the fusion of the bricks forming the floor of the furnace. Such accidents as that are not infrequent in electric furnace experiments when long runs are made.

As an example, a method of constructing a small furnace for heating crucibles, may be described. In Fig. 4 a partial plan, and in Fig. 5 a partial vertical, longitudinal section of such a furnace is shown. In building the furnace the terminals *T* are first set up. These may consist of two carbons of square cross-section with a copper plate clamped (clamp not shown in figure) between them, to make contact with the cables carrying the current. It is preferable to use amorphous carbon electrodes in this furnace, since graphite electrodes will cause a serious loss of heat by conduction. The carbon terminals, as shown in the figure, are needlessly large.

Bricks are then built around the terminal, being cemented carefully with fire clay, so as to keep the carbons protected

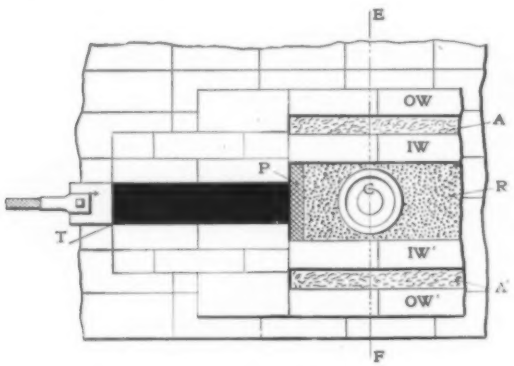


FIG. 4.—PLAN OF ELECTRIC FURNACE FOR HEATING CRUCIBLES.

from the air. During a long run the inner ends of the carbons will become very hot, and, unless the brick work about the terminals is carefuly plastered with fire clay, the carbons are likely to be oxidized.

The inner walls, *IW* and *IW'*, of the furnace are next built up, these being made slightly higher than the depth of the resistor, which consists of granular graphitized coke. In the resistor the crucibles *C* are embedded. Contact between the resistor *R* and the terminals is made by *P*, which consists of graphite powder. The making of this connection is very important, for if not done properly there will be a considerable generation of heat at the point and a consequent waste of energy. Not only does the current pass from a conductor of small cross-section and low resistivity, to a conductor of large cross-section and high resistivity, but even if the terminal had the same cross-section as the resistor, the connection would not be sufficiently good if the grains of the resistor simply rested in contact with the terminal. If an imaginary plane is pictured as passing vertically through the resistor, it will readily be seen that the parts of the resistor on either side of the plane will make much better electrical contact with one another than the resistor would make with a solid slab of carbon against which it rested.

In making this connection in a small furnace, such as is described here, a piece of stout cardboard of the right size should be inserted in the trough about one inch from the terminal, then, as the granular resistor is put into the trough, powdered graphite is put between the cardboard and the terminal and tamped into place till it forms a hard, compact mass. Then more graphite is added and again tamped down, the cardboard meanwhile being withdrawn a little way, so as to allow contact between the graphite and the resistor material. With proper care a most excellent joint is made in this way. It is the kind of joint used by Acheson in the construction of his carborundum furnace.

When the crucibles and resistor are in place the outer walls, *OW* and *OW'*, may be built up, as shown in Fig. 4, leaving

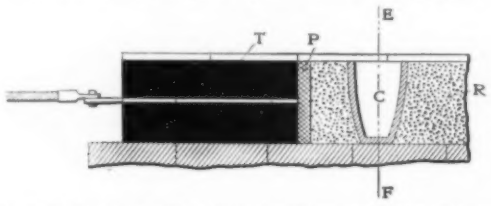


FIG. 5.—VERTICAL SECTION OF ELECTRIC FURNACE FOR HEATING CRUCIBLES.

a space between them and the inner walls. This space is filled with a good heat insulator, such as infusorial earth. If this heat insulation is omitted the losses from heat radiation are serious, as may readily be observed by running such a furnace with and without the outer wall and insulating space. The walls of the furnace, as was pointed out before, are made somewhat higher than the depth of the resistor, so that refractory tile or bricks may be laid over the resistor to diminish radiation losses. In dealing with a large furnace it would be worth while to build a cellular cover to go over the top of the furnace.

If the furnace is to be of a permanent type, it is necessary to construct some form of receptacles for the crucibles. As described, when a crucible is removed from the resistor the granular carbon will fall in and the crucible cannot be returned. It is a simple matter to construct receptacles that fit the crucibles and embed the form in the resistor. These receptacles must, of course, be refractory, and should have as good a conductivity as possible, which indicates the use of silico-carbides for the material.

A furnace built in the way described above will work fairly well, but examination soon shows that its design is by no means perfect. Inspection of Fig. 4 shows that the sectional area of the resistor is least at a plane drawn through the line E F, hence the resistance at this part of the resistor is greatest, and consequently the rate of generation of heat will be greatest, and the heating of the crucible will not be uniform over its whole surface.

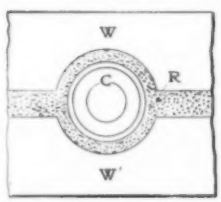


FIG. 6.—PLAN OF RESISTOR.

In order to get even heating, the plan of the resistor should be like that shown in Fig. 6, where the sum of sectional areas of the branches of the resistor passing round the crucible C is equal to the sectional area of the undivided resistor. As far as a horizontal plane drawn through the furnace is considered, this will give uniform heating, but the heating will not be uniform if we consider a transverse vertical plane drawn through E F, for two reasons: First, on account of the shape of the crucible, which gives the resistor a larger cross-section at the base than at the top; second, because the resistivity of the resistor is less at the bottom than at the top. The weight of the upper parts of the resistor compresses the lower parts, diminishing their resistivity. Thus, if we imagine the resistor divided into a series of horizontal layers, the resistances of unit length of the layers will continually decrease as we go lower down. From Ohm's law it may be shown that in a series of parallel circuits the energy developed in each will be inversely proportional to its resistance. In order,

therefore to obtain uniform heating, the cross-section of the resistor should conform to that shown in Fig. 7.

In making a single experiment it is not necessary to go into these refinements of construction, but the conditions may be roughly approximated. If, however, a permanent furnace is to be designed all these details should be kept in mind. But before building the permanent furnace it is necessary to make rough experiments in order to obtain various necessary data for the design. Probably in the future certain general rules and quantities wil be available which will permit of the design of a furnace with the same accuracy that we now design a boiler for a given purpose; but at present this cannot be done, and the experimenter must be content to arrive at his final results by trial. As an illustration of this, let it be assumed that in a certain furnace there are a number of crucibles to be heated; that the amount of energy required for each crucible is known; that the e. m. f. and current available are fixed, then it becomes necessary to determine the sectional area of the resistor and the size of grain to be used. In another article[2] an experiment was described using two granular carbon resistors and data were obtained from which the resistivity of each may be calculated. The resistors were identical, except that one of them (G1) was composed of grains of graphitized coke that passed through a screen having 5 meshes to the linear inch, and were held on a 6-mesh screen, while the other (G2) was made of grains passing through a 3-mesh screen and held in a 4-mesh screen. The dimensions of each resistor, as tested, were as follows:

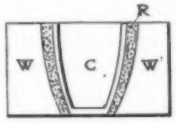


FIG. 7.—SECTION OF RESISTOR.

Length	42 cms.	16.5 inches
Width	5 cms.	2.0 inches
Depth	5 cms.	2.0 inches

Taking the readings at two parts of the experiment, we have the following:

	Volts G1.	Volts G2.	Amperes.
1.	44	30	138
2.	46	32	100

When the first set of the readings given above was taken, weights were resting on the resistors; when the second was taken, the weights had been removed. From these data the resistivities of the resistors may be calculated, and expressed either as the resistance of a cube measuring one centimeter or one inch on the side:

	G1 (cm.)	G2 (cm.)	G1 (inch)	G2 (inch)
1.	0.19	0.13	0.075	0.051
2.	0.27	0.19	0.11	0.075

These figures show clearly the great difference in resistivity found in resistors composed of grains of different size, and also the effect of pressure on the resistivity. The marked effect of the latter becomes still more evident when it is remembered that the area of the resistor to which pressure was applied during the experiment was less than one-third of the total surface area.

For purposes of comparison let us calculate the resistivity of the resistor of a commercially worked furnace with the laboratory experiment given above. Taking the published descriptions of the carborundum furnace[3] we obtain the following data, which are arranged in the form of a table:

Length of resistor (core)	426 cms.
Sectional area of resistor	2210 cms.^{2}
Resistor consists of ordinary coke,	
E. M. F. when furnace at load (746 kw.)	250 volts
Current	2960 amps.
Resistor consists of graphitized coke,	
E. M. F. when furnace at load (746 kw.)	150 volts
Current	4970 amps.
Resistor is now graphitized.	
E. M. F. at end of run	100 volts
Current	7460 amps.

From this table the resistor resistivities have been calculated, the resistivity being the resistance of a centimeter cube:

	Resistivity.
Ordinary coke, on reaching load	0.44
Graphitized coke, on reaching load	0.16
End of furnace run	0.070

The best figures to compare in the experiment and in the carborundum furnace are those of G2 in the former and the end of the run in the latter. It will be observed that they differ considerably, the resistivity of the carborundum furnace resistor being much lower. This is to be explained by the greater size of the coke particles comprising the furnace resistor, for these are from ½ to ¾ inch in diameter; the temperature of the furnace resistor was somewhat higher than that of G2; there is a certain amount of current leakage in the carborundum furnace, and finally the pressure on the furnace resistor is much greater than on the resistor taken for experiment.

The resistivity of the graphitized coke furnace resistor, when it first comes to load, is somewhat higher than that of the resistor G2 when the weight was put on the latter, but at that time it was very hot, while the furnace resistor would be nearly cold.

In the experiment with the resistors, an increase in resistance that was probably due to the formation of carbides was observed, and that enters as another disturbing factor in the comparison of resistivities under various conditions.

This somewhat lengthy discussion of the resistivity of granular carbon resistors seems to be worth while, in order to show the difficulty of the subject, and that our knowledge of it is still very limited. This being so it is not possible to give any definite figures for the calculation of resistors of this sort. It shows, moreover, that the best way of arriving at satisfactory results in the design of such a furnace as has been described is to make preliminary experiments that approximate as closely as possible the conditions that will exist in the furnace. It is of the utmost importance that we study these conditions with great care. Thus, the size of the grains of carbon composing the resistor must be determined by careful screening. A resistor of coarse-grained carbon has, as we have seen, a very different resistivity from one of fine-grained coke, hence the importance of knowing just what is the size of the grains used. Here, too, it may be noted that a resistor containing a quantity of powdered carbon, or fine grains mixed in with the grains that are of the proper size, will cause a lot of trouble, especially in dealing with large furnaces. The reason for this is that the powder will not remain distributed evenly throughout the resistor, and there will consequently be differences in resistivity in different parts of the resistor.

In making the experiments it is important to know the physical and chemical states of the carbon. The resistivities of the carborundum furnace resistors shows plainly the great difference in the resistivities of graphitized and ungraphitized coke. If the resistor used in a furnace is composed of ordinary coke, and its temperature is raised, while in use, to a higher temperature than it has ever been at before, there will be a permanent change in the resistivity of the carbon grains, and consequently in the resistance of the resistor. As may readily be seen, this causes a troublesome irregularity in the works. If, on the other hand, graphitized coke is used, this source of irregularity is eliminated, for in the process of graphitizing, the coke has been raised to the highest attainable temperature and has consequently reached a stable physical state,

[2] ELECTROCHEMICAL INDUSTRY, Vol. II., No. 12 (December, 1904), page 492.
[3] Journal of the Franklin Institute, February, 1897.

Another advantage in the use of graphitized carbon is its purity. In the process of graphitization it has been raised to such a high temperature that a great part of the impurities present have been vaporized. This will naturally diminish irregularities in resistance due to the formation of carbides. It has been demonstrated that silica has a distinctly appreciable vapor tension at temperatures well below that of its fusion, and certain compounds of silicon and carbon are formed at relatively low temperatures. If, then, there is much silica present in the carbon used for the resistor, even if the temperature is not unusually high, there will be a slow formation of carbides which will appreciably effect the resistance.

Keeping all these details in mind, the data for the design of the permanent furnace may be obtained by experiment. The

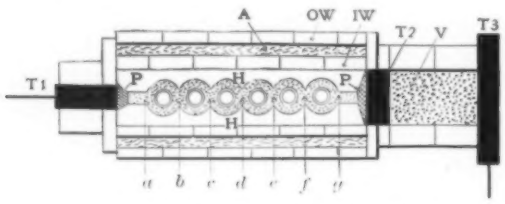


FIG. 8.—PLAN OF FURNACE FOR CRUCIBLES.

permanent furnace will then be similar to the plan shown in Fig. 8. Here $T1$ is a furnace terminal, $T2$ the terminal common to the regulator arc furnace resistors, $T3$ the terminal of the regulator resistor, P is the graphite connection between the furnace resistor and the terminals, A the heat insulator between the outer and inner side walls OW and IW, H a refractory material containing the furnace resistor, V granular carbon forming the regulator resistor. In building the furnace resistor it is best to make a model of one section in wood or plaster, and having built the side walls of the furnace, put the model in place and pack the refractory material round it. It is understood that the refractory material is mixed with some binding substance, so that the model of the resistor can be withdrawn. Thus, the various sections are built in a perfectly uniform manner. The receptacles for the crucibles are put in next, then the granular carbon to form the resistor is introduced and the connections (P) to the terminals carefully made. In putting in the granular carbon care must be taken to avoid ramming it in tightly in one place and leaving it loose somewhere else. It ought to be packed down to a certain extent throughout, otherwise, when in use, it will become more tightly packed in one place than another, through mechanical jarring, and consequently the resistivity will not be uniform. When the resistor is in place the furnace is ready for final adjustment.

Small graphite rods are now thrust into the resistor at a, b, c, etc., and wires attached to the rods are brought to a suitable switching device, so that the rods can be quickly connected with a voltmeter. A small amount of granular carbon is put into the regulator, so that its resistance will be somewhat high and the current is then thrown on. After waiting for a short time the current, as indicated by the ammeter, becomes steady, and then the volts between a and b, b and c, etc., are determined. They will be found to differ, unless the resistor has been put in with absolute uniformity. If it is desired to heat all the crucibles uniformly, the volts across a and b, b and c, etc., must be the same, so now adjustment is made by taking away some carbon where the volts are too low and adding carbon where they are too high. Final adjustment is made by cautiously tamping down those places where the volts are too high. From the preliminary experiments the amount of energy required per crucible has been determined, and, consequently, the resistor must be adjusted to obtain that result.

In making an experimental furnace it is sometimes desirable to have it so arranged that at the same time different parts of the furnace are at different temperatures. A furnace of this kind can easily be built along the lines described above, the resistivity of the resistor being made greatest around those crucibles where the highest temperatures are desired. The best way of arriving at this result is to keep the sectional area of the resistor the same throughout the furnace, and to vary the nature of the resistor material so as to obtain the desired resistivity. Referring again to the experiment with the two resistors of coarse and fine grains, it is seen that where these have been treated in every way the same, G1 has a resistivity of 0.27, G2 a resistivity of 0.19. If, then, one crucible is surrounded with coke, like that used in G2, and the other with coke like that used in G1, the heat developed per second in the latter will be about 37 per cent greater than in the former.

In the description of the crucible furnace given above attention has simply been directed to the general methods of construction, various details being passed over. For example, the question of the material of which the crucibles are made has not been considered. This, of course, will modify details of construction. Thus, if the ordinary graphite or "plumbago" crucible is used, it will be found that it is a tolerably good conductor of electricity, and the furnace will have to be modified accordingly, proper allowance being made for the current carried by the crucible walls. Where the highest temperatures are required the only refractory material that will stand the work is pure graphite. Small crucibles of this kind may be made from graphite electrodes as manufactured by the International Acheson Graphite Co. Here we have, unfortunately, a material that is an excellent conductor for its resistivity is only 0.0008, so that it short-circuits the current at the part of the resistor where it lies. In order to get the most satisfactory heating of crucibles of this sort it is necessary to use currents of great amperage, so that a heating effect is obtained by the passage of the current through the walls of the crucible.

Electrolytic Preparation of Tin Paste.

In the electrolytic deposition of metals it is nearly always desired to get a good, *i. e.*, adhesive and coherent coating. This is always a desideratum in electrolytic refining and electroplating; to avoid a spongy deposit is then one of the chief points the operators have to look out for. A spongy deposit may be due to various causes, but may be obtained nearly always by using too high a current density at the cathode. This results in an impoverishment of the electrolyte near the cathode, with respect to the ions of the metal to be deposited, so that some other metal, and also generally hydrogen, is simultaneously deposited. It is under such conditions that the metal deposit becomes spongy. It is self-evident that under such conditions the ampere-hour efficiency of the deposition of the metal must be relatively low, since part of the current is consumed in setting free hydrogen.

In special cases, where it is desired to get a metal in spongy form, the above rule may be made use of. Such a case is discussed by F. Gelstharp in a recent Faraday Society paper, which we give below. An abstract of the discussion, which followed the paper, may be found on another page of this issue under Faraday Society Meeting.

Tin paste is used for the production of metallic or silvered paper, having the appearance of tin or lead foil, and it is also used for ornamental purposes. The process for manufacturing tin paste, described below, was specially designed for a firm of paper makers.

The oldest method of making tin paste was that carried out by the natives of India. They melted block tin at a high temperature, and poured it into a wooden box, which was then shaken vigorously; the tin was thus converted into very fine shot and dust. The finer particles which they used for

the paste were separated from the coarser granules by gravitation by means of water. It was afterwards dried and mixed with a weak solution of glue, and in this state it was ready for use. This they used as a paint, the article to be ornamented was coated with it, and after being dried it was burnished.

The chemical process for making tin paste, as generally carried out in England, is as follows: The commercial ingot is reduced to what is known as feather tin by melting it and then allowing it to fall some distance. The tin in this form is dissolved in strong hydrochloric acid, with the aid of heat, in copper vessels, and a concentrated solution of stannous chloride thus obtained. The liquid is diluted and the tin precipitated in the form of paste by introducing metallic zinc. The process is troublesome on account of fumes, and, moreover, does not give the whole of the precipitated tin in the form of paste, but partly as bright metallic feathers and crystals, which have to be worked over again; whereas in working the electrolytic process about to be described there are no fumes, and the paste is perfect in every respect. Further, the process is not so costly and can be worked continuously, the deposited paste being removed at intervals without stopping the current.

The electrolytic process is similar to the electro-refining of metals. Anodes of tin are dissolved in an electrolyte of dilute hydrochloric acid and the tin is deposited in the form of a sponge or paste on cathodes of the same metal. The spongy deposit is lighter than the electrolyte on account of the hydrogen it contains, and consequently floats on the surface of the solution after being detached from the face of the cathodes, so that it can be conveniently removed without breaking the current.

An apparatus capable of producing 20 cwt. (2000 pounds or 900 kg.) of tin paste per week, consists of a generating plant giving 1500 amperes at eight to ten volts. There are five vats, which are worked in series and are made of slate or pitch-pine wood 106 × 106 × 91.5 cm. deep (3 feet 6 inches × 3 feet 6 inches × 3 feet deep), each containing four anodes and five cathodes coupled in parallel.

The anode plates are about 2.5 cm. (1 inch) thick, roughly cast from commercial ingots of tin; the surface exposed to the electrolyte being 91.5 × 76 cm. (3 feet × 2 feet 6 inches). The cathode plates are of block tin or tinned iron, the active surface being 91.5 × 76 cm. (3 feet × 2 feet 6 inches).

The anodes are supported at the two top corners, and in each vat are all connected to one conductor. The cathodes are supported by and soldered to copper conductors, which are all connected to a main conductor. These main conductors are of circular copper bars 3.2 cm. (1¼ inches in diameter).

The electrolyte is a dilute solution of hydrochloric acid in 5° Tw., which must be free from arsenic. The current density is 270 amperes per square meter, or 25 amperes per square foot of cathode surface.

As the anodes are dissolved away they are removed before they crumble and fall to pieces, and what remains of them is melted up with fresh ingots and cast, to form new anodes.

The electrolyte is kept in circulation by means of a small turbine or plunger pump of lead or block tin driven by a small motor, the liquid is drawn from the bottom of vats and discharged into the top; this is found necessary, as otherwise the electrolyte at the bottom of the vats would soon become very rich in tin, and instead of tin paste at the lower parts of the cathodes there would be crystal or leaf tin deposited.

When the deposited tin sponge has grown to about an inch in thickness it is detached from the cathodes by means of a T-shaped scraper, which is pushed down the face of the cathode plates; the floating, spongy tin is removed from the surface of the vat by means of a perforated scoop to prevent short-circuiting of the current through the floating tin; the cathode plates are varnished or painted with a non-conducting material from 5 to 7.5 cm. (2 or 3 inches) below the surface line. The tin paste is allowed to drain, and is then thoroughly washed with water and dried to make it ready for use.

This may not seem to be an efficient way of utilizing the electric current, because the current efficiency is only about 50 per cent; but any attempt that I have made to produce tin in the form of paste or sponge at a higher efficiency has been futile; it appears to be necessary to have hydrogen deposited along with the tin, for when no hydrogen is deposited the tin comes down in a crystalline or bright leaf-like form.

Some Present Problems in Technical Chemistry.*

BY PROF. W. H. WALKER, PH. D.

(*Concluded from page 28.*)

In an address on Chemical Problems of To-day, delivered by Victor Meyer in 1889, the author pointed out that, although the synthesis of starch from carbon dioxide and water was a result not to be expected in the near future, yet, he says, "we may reasonably hope that chemistry will teach us to make the fibre of wood the source of human food." While we do not consider that this is a problem of technical chemistry for the present, the possible use of cellulose as a raw material from which to make food, renders more acute a problem which is to-day clamoring for solution, namely, the preservation of our forests. The influence which the forests of a country have upon its civilization is a topic which has been much discussed of late. That there is an intimate relation between the woodland of a district and the regularity of its rainfall, the absence of floods and freshets and the general climatic conditions, there seems now to be little doubt. But the consumption of forest products continues to increase far out of proportion to the growth of new timber. The substitution of other raw material in chemical industries which now use wood for this purpose becomes, therefore, an economic problem for the solution of which the chemist is held responsible.

The production of cellulose from raw materals other than wood is the first important factor in the chemical side of the question. The weight of wood consumed for the production of chemical fibre for the year 1902 was something over 2 million tons, while 1½ million tons were used for the manufacture of ground wood pulp. While from some points of view our American forests are sufficient to supply the demand for many years to come, it does not excuse us for the terrible waste of cellulose in forms other than wood, which we are constantly suffering.

On our flax fields of the West we are annually burning thousands of tons of flax straw which contains a large percentage of cellulose in a most valuable form. Considerable work has already been done on the utilization of this straw in the production of fibre, and some success has met the efforts of the By-Products Paper Co., now located at Niagara Falls. There is, however, still much room for improvements. In the straw of our wheat and oat crops, which is to-day largely destroyed on the fields, we have another source of cellulose of which we avail ourselves but little. In Europe the production of straw fibre is carried on to some extent, but is capable of great extension should sufficient economy in the process for treating it be introduced. The high content of silica has ever been a source of loss, owing to the fact that the formation of sodium silicate prevents the recovery of the soda now used in the digestion of the straw.

By far the greatest loss of valuable cellulose, however, is found in waste cornstalks and in bagasse or the sugar-cane after the soluble portions have been removed. There is a

* An address delivered before the International Congress of Arts and Sciences at St. Louis.

close analogy between these two products, in that there is associated with the woody portion carrying the cellulose a large amount of non-usable pith. Rapid progress has been made in the utilization of both of these raw materials within the last few years, and the indications are that before long they will prove a source of value rather than a nuisance, as is frequently the case at present. The market price of bleached cellulose fibre is to-day from 2½ to 3½ cents per pound. Starch may be bought for from 2½ to 4 cents, according to its source. It is seen, therefore, that there is little manufacturing margin in the conversion of cellulose to starch or sugar until the cost of the former has been considerably reduced. This can come about only through new processes designed to operate more economically than those at present in use, and to use as raw products the cellulose at present wasted on the fields.

It would seem that a more economical step toward the production of food from wood might be through its ligneous or non-cellulose constituents. For every ton of cellulose produced there must be used two tons of wood; that is, an equal weight is wasted. In the soda process, as now conducted, these non-cellulose materials are burned to recover the soda which is held in combination with them. In the sulphite process this enormous amount of material, aggregating for America alone in a single year almost one million tons, finds its way into the water courses and ultimately to the ocean. This organic matter is most complex in its composition, but consists largely of one class of substances closely allied to the sugars, and another class having the general characteristics of tannins. That these sugar-like substances could be made to yield a food material is, from their nature, quite possible; so far as we know, however, but little has been accomplished in this direction. A number of uses have from time to time been proposed for this waste, but as yet none have been of practical value. Among the more promising may be mentioned a preparation to be used in tanning leather, a sizing material for paper and a substitute for dextrine in calico printing, and as an adhesive.

In addition to our annual supply of 4,000,000 tons of paper stock, we depend upon the forests for our supply of acetic acid, methyl alcohol and acetone. In countries where there is not the exorbitant tax upon fermented mash that exists in the United States, there would seem to be an opening for a process for the production of acetic acid from alcohol in a more concentrated form than can be produced through the aid of mycoderma aceti. It would, it is true, in the end depend upon the supply of fermentative material; but there are being wasted every year in the semi-tropical countries many thousand tons of crude molasses that could thus serve an economic end. For many uses acetic acid may be displaced by formic acid, a compound which admits of synthesis from carbon and water. The farther this substitution is carried the more acetic acid will be available for the manufacture of acetone and other compounds where the acetyl group is a necessity.

Concurrent with the disappearing forests is the increasing scarcity of vegetable tanning material. Hemlock and oak bark, sumac and chestnut wood are still the most important sources of tannins, although quebracho from South America and canaigre from Mexico and Texas are daily playing a more imporant part. The introduction of chrome tannage for upper leathers had a marked influence upon this industry, inasmuch as it furnished a cheap substitute for those finer tanning materials which are constantly increasing in price. A mineral tannage for heavy hides, along the lines so successfully followed for upper leather has, however, not been developed; the product lacks the rigidity and firmness combined with the flexibility which is characteristic of oak or hemlock tanned leather. There must exist methods for supplying to the hide materials having an action analogous to these vegetable tannins; it remains but to seek them out in order that a new and profitable industry may be established.

It is thus seen that technical chemistry can do much for the conservation of our forests; along many lines the time for action has already come.

When the consumption of a given article is in excess of its supply, the market price must rise. In accordance with this law we have seen the price of crude India rubber more than double in the last few years. The consumer of the finished article must pay this advance or accept an inferior grade of goods. Generally he does both.

The tropical forests of Africa and South America still contain untold quantities of India rubber; but so does sea water contain gold. For manufacturing purposes both might as well not exist. The only human beings that can live under the conditions obtaining in these tropical jungles are the natives; but the distance to which the natives can transport the rubber is comparatively limited. Although rubber-bearing trees are now being cultivated in the more easily inhabitable portion of the tropics, it will be a long time before the source of supply is an important factor in the market. And thus it comes that the synthesis of India Rubber presents today from at least the technical side, one of the most promising problems in chemistry.

The investigation of India rubber is greatly handicapped by the fact that it exists only in the colloidal state. The difficulties are, perhaps, more largely physical than chemical; that is, it is the molecular aggregation rather than the atomic structure of the individual molecule which presents such almost insurmountable difficulties. There are no clearly defined melting points, boiling points, tendencies to crystallize or any of those means of separating mixtures or characterizing individuals which aid in the investigation of most organic compounds. The researches of Weber and Harris, resulting in the establishment of the much-needed methods of analyses, have been of incalculable advantage to all those working with either the raw or the manufactured article. In many directions also, the paths along which important results are to be obtained have already been blazed by these investigators. Probably no other field presents such difficulties of manipulation, in addition to such profound problems of organic chemistry as does the investigation of India rubber; but on the other hand, few such unlimited opportunities for valuable work are offered in the field of chemical research.

Under the general head of utilization of trade wastes may be considered a large number of technical problems, the solution of which would not only add wonderfully to the economic resources of the country, but would aid in the solution of that much vexed question, river pollution. We have already mentioned the soda and sulphite liquor resulting from the manufacture of cellulose fibre from wood. Of almost equal importance is the waste yeast which is daily produced in the brewing of beer and ale. An extract of this yeast has a food value, as shown by analysis, equal to the best meat extracts. As the quantity of yeast allowed to go to waste is from one to two pounds for every barrel of beer brewed, we can form estimates of the great amount of this material at hand. Arsenic sulphide from the purification of crude acids, grease from the washing of wool, the utilization of city garbage and many other problems of this order are everywhere in evidence. It is not within the compass of this discussion to mention these almost innumerable sources of manufacturing waste which exist in the chemical industry; but keen competition on the one hand, and the State Boards of Health on the other, are constant stimuli to increased effort toward their utilization.

Although I have endeavored to select the above examples of unsolved problems with a view to touching upon as large a portion of the field of Technical Chemistry as possible, I could doubtless, with equal propriety have selected others. We can simply mention such important questions as the hygienic

preservation of food, the flame-proofing and preservation of wood, prevention of the corrosion of structural iron and steel, the great problems of chemcal metallurgy, et cetera. We must, however, note some of the more recently developed forces and phenomena of nature, the application of which to technical chemistry forms problems for to-day. One of the most important of these is electricity. Thanks to the triumphs of modern electrical engineering we are now able to call to our aid unlimited amounts of this agent at a cost comparable to that of other forms of energy. Possibly the simplest, though not the earliest method of utilizing electrical energy in chemical processes is in supplying the heat necessary to carry on a reaction directly at the point where the reaction takes place. In a number of chemical industries, for example, the manufacture of phosphorus, it was previously necessary to produce within thick-walled retorts a very high temperature. The result was that a great deal of heat was wasted, the retorts deteriorated very rapidly and the reaction was carried on at a low efficiency. By using an electric furnace for the manufacture of phosphorus these expensive retorts are eliminated. In addition much cheaper raw materials may be used, the process is made continuous and a high efficiency obtained.

By the substitution of electrical heating for the closed retorts previously used in the preparation of carbon bisulphide the manufacture of this chemical has been placed upon an entirely new basis. The economy introduced by supplying the heat at the point where the union of carbon and sulphur takes place is clearly indicated by the low price at which this material can now be sold and its enormously increased consumption.

With the ability to obtain temperatures far above that which is possible by the ordinary combustion of fuel, there was opened up a new field in synthetic chemistry. Reactions which it was impossible to carry out on a technical scale, and others, the existence of which was not suspected, have now, through the application of electrical energy, become the bases of large manufacturing enterprises. Calcium carbide, carborundum, artificial graphite and many hitherto unknown alloys are the commercial products of the electric furnace where temperatures in the neighborhood of 3000° C. obtain.

The third and more strictly chemical application of electrical energy is in the use of the current for electrolysis. Faraday long ago determined the laws according to which chemical compounds break up when subjected to the passage of an electric current. It is only in recent years, however, that the cost of electrical energy has made it possible to apply the knowledge thus furnished by this great investigator. Among the many important advances due to this use of electricity may be mentioned the manufacture of caustic soda and bleaching powder by the electrolysis of brine. The percentage of the world's supply of these two standard articles, which is now made by this process is already a formidable figure, and constantly increasing. In the electrolytic production of aluminium we have seen an entirely new industry develop, until it is now one of magnificent proportions.

What the application of electricity will do for technical chemistry in the future can be predicted only by estimating the results of the past. In many fields it is practically virgin soil over which only the pioneers have trod, and which is still waiting to be tilled.

Under the name of catalysis or contact action is included the other force that we can mention this afternoon, the usefulness of which the technical chemist is only beginning to appreciate.

These substances, which are capable of so wonderfully increasing or decreasing the speed of a reaction without themselves appearing in its final products vary in their nature from such simple ones as metallic platinum or ferric oxide to the most delicately constituted ferments or enzymes. The manufacture of concentrated sulphuric acid by such a process is, perhaps, the most striking example of the application of this idea, although, to be sure, the finely divided platinum used at present plays but the role which the oxides of nitrogen have done so successfully in the past. The reproduction of photographic negatives by substituting for the action of light on sensitized paper the contact action of certain chemical compounds, is a process worthy of its distinguished discoverer, Professor Ostwald. For this application of the catalysis idea even the most pessimistic must prophesy a great future. Still another phase of this question is found in the hydrolysis of fat by the enzyme found in the seeds of the castor-oil plant. Instead of the application of acid, heat and pressure the same result is obtained at room temperature by the quiet action of this catalytic body. The advantages to be reaped by the development of these phenomena can scarcely be foreseen. Even the wildest dreamer might easily do justice to the possibilities of this wonderful agent when intelligently used by the technical chemist.

We probably would not invite criticism were we to state that wherever we find a manufacturing establishment based upon chemical processes, there also exist problems in technical chemistry. That one factor which is so apparent that it scarcely needs mentioning, namely, the increase in the yield of processes now in operation, is enough to substantiate this assertion. The paramount question before us is, therefore, how can these problems best be solved. In any answer to this question there are two factors, both of which deeply affect the future growth of chemical industry. The first is the attitude of the manufacturer towards science and scientific work; the second is the training of the coming chemist. When, a few years ago, England awakened to the fact that many industries in which she was the pioneer and at one time the leader were in the main passing to other countries, there went up a great cry for "technical education." The nature of the industrial stimulus which has borne such magnificent fruit in Germany was not understood. In the minds of many, a panacea for all their difficulties was to be found in the technical education of the working classes. But this is unquestionably a mistake. Until there is a love of science for its own sake and an appreciation of the value of scientific method among the leaders of chemical industry, the fruits of technical education cannot be reaped. Carl Otto Weber, speaking of this move towards a more general scientific education in England, says: "Until the nation, as a whole, recognizes that the prosecution of scientific study as a mere means of money making is a profanation defeating its own end, the history of industrial developments in England will afford the same melancholy spectacle in this, as in the last century, technical education notwithstanding."

The time is past when a factory can be run by rule of thumb; when the chemist is looked down upon simply as a testing machine to be kept at a distance and generally mistrusted. It is true that there are many men to-day who pass under the name of chemists who are little more than testing machines; men who possess the ability to do nothing more than the most strictly routine analysis; but such men will never solve the technical problems of the present or any other time. I would not impugn the dignity or intrinsic value of analytical work—it is the corner-stone of all chemical investigation. But I would emphasize the fact, for it is a fact, that the manufacturer who employs a so-called chemist, one trained to "do" coppers or carbons or acids, and who at the same time expects this chemist to improve his process and keep his business in the skirmish line of the industrial battle, must eventually be numbered among the "not accounted for."

The second factor in this answer is the training of the coming chemist. What is the reply to that now so oft repeated question: What is the best preparaton for a technical chemist? I am personally of the opinion that it is not to be found in the teaching of applied chemistry, as this term is generally understood. This training must provide for something more than simply copying the present—doing as well as others do;

we must build for the future. We must provide men who are prepared to solve the unsolved problems. Within the last few months much has been said and written in America about the lack of adequate instruction in technical chemistry in our universities and colleges. It is assumed that American industries, based on chemical processes, do not flourish for lack of men trained in this branch of science. This, however, is not the case. It is not more instruction in applied chemistry that America needs, but rather a deeper and broader knowledge of pure chemistry, with a more extended training in original research.

In many of the problems we have already noticed, the solution depends upon the discovery of new compounds—the investigation and study of new reactions and relationships. This is the province of pure organic and inorganic chemistry. The foundations of these two departments cannot be too firmly or too broadly laid. The method of attack best followed in each cannot be too well understood. But it is not sufficient that we study only the initial and the final products. It is all important to learn the influence of the variable factors on the process; to study the reaction for itself. This is the province of physical chemistry, a department of science, the importance of which, to technical chemistry, cannot be overestimated. To be able to actually apply the laws of chemistry and to predict the course of reactions from general principles already proven is a tremendous economy of both time and energy.

After we have acquired the tools, however, we must learn to use them; after we possess a sound knowledge of inorganic, organic and physical chemistry, we must have adequate training in work requiring original and independent thought.

As I have already noted, the training to be derived from an investigation may be the same even though the incentive for its undertaking may be different. While I believe that so far as possible the student should be influenced to work for the love of knowledge and for the mastery of science for itself, yet especially in his later years of study there are advantages in allowing him to combine with this a utilitarian aim. In America, at least, most men enter our technical schools with the intention of fitting themselves as rapidly as possible for some useful calling in life. They have a feverish desire to get through and to enter the creative industries and accomplish something. They will work with enthusiasm upon whatever they can be made to recognize as contributing to this end, but by their very directness are intolerant of supposed digressions from their chosen path. The presence of too much of this spirit is to be regretted; but it is a power to be turned to service, not to be opposed. It does not follow that for a training in scientific method and for broadening the mental horizon a research which can have little, if any, practical value is superior to one, the solution of which can find immediate application. For advanced work, as much pure organic chemistry, for example, can be learned from an attempt to convert safrol into eugenol (a consummation in itself devoutly to be wished), as in the transformation of some other compound with a much longer name, but with no higher destiny than to fill a place in Beilstein.

So, also, in physical chemistry. A careful, painstaking investigation of some of our already established industrial processes with a view to determining the maximum yield at the minimum cost is of the greatest educational value. In other words, a problem for research may have a distinctly practical bearing without being any the less a study in pure science, or without having thereby an inferior educational value.

In other problems we have noted, the solution largely depends upon the process, not the reaction. This demands the chemical engineer, a man who combines a broad knowledge of general chemistry with the essentials of mechanical engineering. He must be well schooled in the economics of chemistry; have a knowledge of the strength and chemical resistance of materials; be able to design and operate the mechanical means for carrying out on a commercial scale the reactions discovered, and duplicating the conditions already determined.

With men whose foundations are thus broadly and deeply laid, anxious to enter the industrial arena; and with a generous appreciation of the scientific man on the part of the manufacturer, coupled with a willingness to grant him an adequate return on the money invested in such an education, the problems in technical chemistry of the present must rapidly become the achievements of the past.

Electrolytic Recovery of Tin.

The electrolytic detinning of tin plate (tinned sheet iron) has become an important industry during recent years. In Germany alone, about 35,000 tons of tin scrap are treated annually, at present, by electrolysis, from which 2 to 2.5 per cent, *i. e.*, about 800 tons of tin, are recoverd, amounting to about $500,000. In spite of this extent of the industry, little exact and authoritative information on the details of the processes is available, since the most successful companies in this field keep their methods carefully secret. In view of this situation, any information on experiments in this line requires attention. A recent Faraday Society paper of F. Gelstharp, printed below, is interesting in this respect. Some remarks, made in the discussion of this paper, are given on another page of this issue under Faraday Society Meeting.

An experiment was made by Mr. Gelstharp with an electrolyte of pure sodium hydrate in water, strength 10° Tw. (sp. gr. 1.05), temperature about 80° C., anode and cathode of tin plate, area 19½ sq. cm. (3 square inches), placed about 4 cm. (1½ inches) apart.

After switching on the current a small quantity of hydrogen gas was liberated at the cathode, and a little oxygen at the anode, gradually diminishing. The voltmeter stood at 3½ volts, and the ammeter at one-third ampere. The current was allowed to pass for about half an hour to see if the usual energetic electrolysis would commence; but no, the current continued to effect only a slight evolution of gas without stripping the tin from the anode. On examining the anode it was found to be very little attacked, but appeared to be covered with an extremely thin coat of a light brownish tint, which was not easily rubbed off. The current was again turned on without any different result.

The coloration was thought to be due to some low oxide of tin, which is a poor conductor of electricity. Therefore, the effect of reducing it by means of hydrogen was tried, this being done by reversing the current for a moment. Then, after reversing again the direction of the current, the tin-plate anode was completely stripped of its coating of tin in a very few minutes, hydrogen gas being energetically liberated at the cathode, which also became coated with loose, spongy tin. While this reaction was going on the ammeter rose to 1½ amperes, and the voltmeter fell to less than one volt, and when the cathode was agitated so as to knock against the sides of the containing vessel, the ammeter rose to two amperes, and the voltmeter fell to about half a volt.

Various pieces of tin plate were tried under the same conditions, and it was noticed that the clean, bright pieces acted in the same way as that just described; but pieces of old, oxidized, and tarnished tin plate were attacked at once, and quickly stripped.

It was found that instead of reversing the current to make the reaction start, it could be brought aobut by placing in contact with the anode in the electrolyte for a short time a piece of clean iron, most effective being a piece of tin plate which had just been stripped in the bath.

From the results of these experiments the writer conjectures, or suggests, that the clean, bright tin plate (differing from tarnished or oxidized tin) at first becomes thinly

coated with a black or brown oxide which is insoluble in sodium hydrate, and that the effect of reversing the current, or making contact with iron, causes a deposit of hydrogen on the anode, which reduces this oxide to metal, thus roughening the surface, and making it in a favorable state for forming the soluble oxide, and when once the formation of soluble oxide has commenced, the electrolysis goes on uninterruptedly. I bring this reaction before the society in the hope that some member who may have investigated it will endeavor to give a satisfactory explanation of it.

After all, the tin was stripped from the anode, the ammeter again fell to about half an ampere, the voltmeter rising to 2½ volts; and when the current was continued the iron became coated with a reddish-colored deposit (probably composed of oxide of iron), which was not easily rubbed off. I noticed that the deposited tin, when squeezed and placed in water, generated hydrogen gas. This evidently points to the fact that spongy tin deposited in a sodium hydrate electrolyte is not only an alloy of tin and hydrogen, but also contains some sodium.

The agitation of the cathode in a special way seems to be an important point so far as the consumption of electric energy is concerned. The resistance due to polarization can be reduced to a minimum, not only by moving the electrode vigorously and by quick circulation of the electrolyte, but by giving it a quick succession of shocks produced by striking sharply. By this means the electric energy can be reduced by 30 to 40 per cent.

Comparing the acid and alkali processes for tin recovery, it is beyond doubt that the latter process is superior. In the alkali process iron apparatus may be used, and mechanical appliances for continuous treatment of tin-cuttings can be conveniently adapted; moreover, the electrolyte is continually being regenerated. In the acid process no suitable metallic apparatus can be used; therefore, mechanical appliances are hardly admissible, and for the electrical treatment of tin-cuttings to be a commercial success they must be handled very cheaply. This can only be done by mechanical means with a continuous process.

For the manufacture of tin paste the acid is superior to the alkali process, as the deposited tin from the former is very light and amorphous, whilst that from the latter is much denser, and consists of minute crystals, which are not so suitable for the uses to which this material is put.

Chloro-Electrolytic Smelting.

We have repeatedly referred in our columns (especially Vol. I., page 412, and Vol. II., page 404) to the Swinburne-Ashcroft process of treating complex sulphide ores. The process consists essentially of three steps. First, treatment of the complex sulphide ores at a temperature between 600° and 700°, with chlorine, with a resulting change of the mixed sulphides into chlorides, sulphur being set free; this step, which is purely chemical, is carried out in what is called the transformer. Second, substitution of one metal for another in the mixed chlorides, so that there is finally only zinc chloride left; this step, which is also purely chemical, is to be sub-divided into as many partial substitution processes as there are metals besides zinc present. Third, electrolysis of zinc chloride, yielding zinc and chlorine. The chlorine is used again in the transformer for treating new quantities of sulphide ore. The diagram given in our Vol. I., page 412, shows clearly that if the process is carried out with theoretical perfection, according to the above scheme, all the metals and the sulphur contained in the original sulphide ore are recovered separately, while the process is cyclic with respect to chlorine, that is, the chlorine is used over and over again.

The first two steps only of the process are applied by the (British) Castner-Kellner Co. (our Vol. II., page 404). The reason for this departure from the original scheme is that this company has a supply of electrolytic chlorine on hand, and wants to use it for the production of zinc chloride. The starting materials in their operation are therefore complex sulphide ore and electrolytic chlorine, the end products zinc chloride, sulphur, and all the metals—except zinc—which were contained in the sulphide ore.

What appears to be the most complete treatment of the whole process, is contained in a paper, recently presented before the Chemical Society in London by Prof. W. MORLEY COBELDICK, of the Royal College of Science. This paper discusses fully the application of the Swinburne-Ashcroft process to the treatment of Broken Hill ore, which may be described as a mixture of PbS, ZnS, Ag_2S, FeS_2, MnS and gangue. It is ground to a coarse powder, dried and transferred to the transformer, where it is decomposed by the chlorine; this results in a fused mixture of:

$PbCl_2$, $ZnCl_2$, AgCl, $MnCl_2$ and $FeCl_2$,

with the gangue held in suspension. The sulphur vapor is drawn out of the transformer and condensed. This is the first step.

In the second step, silver, lead, iron and manganese are consecutively removed by substitution, as follows. When the transformer is full of chlorides, they are placed in the "desilverizer" where the silver is removed by agitating the fused chlorides with molten lead, the following reaction taking place:

$$2AgCl + Pb = 2Ag + PbCl_2.$$

The desilverized chlorides are then transferred to the "deleader" for stirring them with sufficient molten zinc to cause the reaction:

$$PbCl_2 + Zn = ZnCl_2 + Pb.$$

The resulting chlorides contain only:

$ZnCl_2$ $FeCl_2$. $MnCl_2$, and gangue.

In order to separate the gangue matter it is necessary to dissolve the chlorides in water, and advantage is taken of this to precipitate the iron and manganese as hydrated oxides. To effect this replacement there are several ways of proceeding, but the following is the simplest: A current of chlorine is sent through the liquid which oxidizes the ferrous and manganous chloride to ferric and manganic chlorides. Crude zinc oxide is stirred in, and the hydrated oxides are precipitated:

$$Fe_2Cl_6 + 3ZnO + 3H_2O = Fe_2(HO)_6 + 3ZnCl_2$$
$$MnCl_4 + 2ZnO = MnO_2 + 2ZnCl_2.$$

This result may be brought about by using other materials, but in all cases zinc must be added in some form in sufficient quantity to combine with the chlorine of the iron and manganese.

The liquid now contains:

In suspension: All the iron, manganese and gangue originally in the ore.

In solutions: Pure zinc chloride containing all the chlorine used to decompose the original sulphide, combined with the zinc equivalent to all the metals in the sulphide.

It is filtered or allowed to settle and the clear zinc chloride solution evaporated till the chloride fuses, when it is electrolyzed, chlorine and zinc being the products.

TRANSFORMER.

The "Transformer," which is illustrated in Fig. 1, consists essentially of a sheet-iron vessel lined inside with firebrick, or other heat and chlorine resisting material. The chlorine is injected at the bottom, the crushed ore is fed in at the top, and sulphur vapor passes out from a pipe in the top and is conducted to condensers. Tapping spouts are provided for running out the fused chloride after the completion of the charge; they are so arranged as to leave behind sufficient molten chlorides for starting the next charge of ore.

Very little is known as to the reactions which take place in the transformer. Sulphur chloride, and ferric chloride are possibly formed, but at the temperature of the transformer it

immeditely breaks down into sulphur, which is given off in the free state, and chlorine, which combines with the metals.

Ferric chloride, if formed, would immediately be reduced to ferrous in the presence of the sulphide ore. The fact that it is possible to obtain chlorides of the metals and sulphur by treating sulphide with S_2Cl_2 or Fe_2Cl_6 at 600° to 700° indicates that some such action as the above is possible.

The management of the transformer is exceedingly simple and can be mastered by an intelligent laborer in eight hours. Thus, if the temperature falls below the prescribed limit, sulphur chloride begins to form, and in coming over with the sulphur vapor alters the color of it. Its presence is also indicated by its characteristic odor, and consequently the man has to increase the speed of working by adding more ore and turning on more chlorine. If the temperature rises above the limit, the sulphur vapor is colored white, owing to the zinc chloride commencing to distil over. This is a sign to decrease the speed of working. When the whole of the sulphide is decomposed, the fact is proclaimed by the production of chocolate-colored vapor of ferric chloride, and the sulphur vapor ceases to come off.

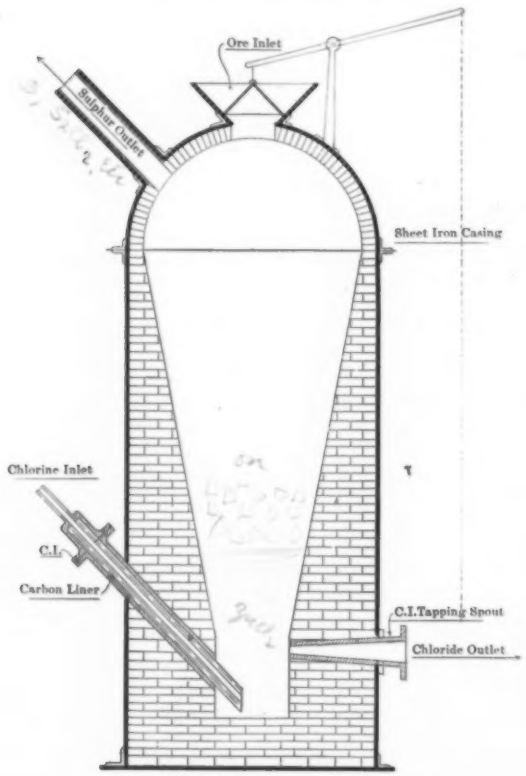


FIG. 1.—CONSTRUCTION OF TRANSFORMER.

The transformer reaction is strongly exothermic, so that no external heat is required to continue the reaction, the temperature being governed simply by the rate at which chlorine is passed through the fused mixture. It seems impossible to pass the chlorine so quickly as to obtain it passing out with the sulphur vapor. Immediate absorption takes place and the only effect of passing the chlorine rapidly is to raise the temperature. It is possible to work so rapidly that the chlorides are raised to their distilling temperature—in fact, with ores very rich in sulphide the rate of working depends on the rate of cooling of the walls of the transformer. Galena is a notable example of this kind of ore.

The efficiency of the transformer operation when properly carried out, amounts to about 98 per cent. If the chlorine gas supplied to the transformer is perfectly pure, all the sulphur is recovered, but this never happens in practice, since the chlorine invariably contains a small quantity of air which causes the formation of a little sulphur dioxide.

The fused chlorides are very fluid and can be easily handled, iron ladles being used to carry them about. The fluidity depends on the amount of gangue present, and it is necessary to keep the proportion within limits. The maximum percentage of gangue in a chloride which is workable is 30 per cent.

Condensation of the Sulphur.

For condensing the sulphur two methods are employed. To make flowers of sulphur the vapor from the transformer is caused to pass through a brick chamber, which is provided with a partition up the center extending from the tap down to within a foot or so of the bottom. The vapor enters one side at the top, travels down to the bottom, and finds its way through the space at the bottom of the partition into the other side, where it rises to the top and escapes by an outlet connected to the works chimney (with a sulphur dioxide absorption tower, if necessary). The sulphur condenses on the walls and is cleaned out periodically.

To make roll sulphur, the vapor is led to a large cast-iron vessel made in two halves. The lower half is set in a brick furnace, and is fitted with tapping spouts. The upper half is exposed to the air and contains openings to allow the vapor to enter and the waste gases to escape. The sulphur condenses in the top part, and owing to a small fire kept in the furnace, it gradually melts and falls to the bottom, where it accumulates and is periodically removed.

The Desilverizer.

The desilverizer consists of a cast-iron pot fitted with a cover, and from this is suspended a cast-iron agitator, which is worked by the gearing on top. The post is set in a brick furnace, in which sufficient fire is kept to maintain the chlorides and lead in the molten state. The silver alloys with the excess of lead in the pot, and this lead-silver bullion is used for desilverizing until it reaches a value of 1,000 ounces per ton. The extraction of silver is quite complete, the operation usually lasting about one hour and a half.

The Deleader.

The desilverized charge runs direct to the deleader, which consists of an agitator pot similar to the desilverizer. It is arranged in a brick furnace in such a manner as to have very close control over the temperature. A gas fire is to be preferred on account of the greater ease with which the temperature can be controlled. The reason for this restriction is that, unless the chloride is kept at practically a constant temperature, while the zinc is being stirred into it, there is a great risk of the ferrous chloride being reduced to metallic iron, which forms a triple alloy of iron, lead and zinc, very hard and infusible at even a bright red heat.

The deleading reaction is exothermic, and on starting the charge, the heat given out is sufficient to maintain it in the molten state, but as the proportion of lead decreases, the temperature would fall, unless maintained by externally applied heat. The fire has, then, got to be very small at first, gradually being increased as the lead percentage falls. The operation of deleading takes about two hours.

Dissolving the Chlorides.

From the deleader the molten chloride is run into water contained in a wooden tank fitted with an agitator. The zinc, iron and manganese chlorides dissolve and the gangue remains insoluble. The temperature is raised to about 70° C. by the introduction of the fused chloride. Chlorine is blown into the liquid, which is kept mixed by revolving the agitator.

The zinc oxide is obtained by using a cheap form of zinc oxide ore, which is ground and introduced gradually into the liquid.

There are many other ways of precipitating the iron and manganese—zinc oxide and potassium chlorate; bleach powder; lime and chlorine being a few of the oxidizing and precipitating agents which may be employed. In the case of lime, it is necessary to replace the calcium chloride, which is formed by zinc. This is done by adding sufficient zinc sulphate to precipitate the calcium as sulphate, zinc chloride going into solution.

When the whole of the iron and manganese is precipitated the liquor is pumped either into a settling tank or through a filter press. It is found that the best results are obtained when the zinc chloride solution is about 1.48 Sp. G. (= 45 per cent $ZnCl_2$). This is the strength used in practice. The solution of zinc chloride is run from the filter press, or settling tank, into the evaporator. The gangue and oxide is washed free from zinc chloride, and the washing water used for dissolving the next batch of deleaded chloride.

An alternative method of proceeding, which omits the risk of iron being precipitated with the lead, is to dissolve the desilverized chloride in water. The iron and manganese is precipitated as hydrated oxide and the liquid allowed to cool It is then passed through a filter press, and the purified zinc chloride forms the filtrate, which is evaporated and fused.

The lead chloride, on account of its sparing solubility in the cold solution, is practically all retained in the filter cakes, and is extracted by leaching these in the press with boiling concentrated brine. The lead chloride readily dissolves in this at 100° C., but separates out again on cooling. To effect this the brine is run from the press to cooling tanks, and the lead chloride separates out, while the brine is used again for leaching another batch of cakes. The lead chloride is dried in a centrifugal separator, melted and deleaded with metallic zinc. Anhydrous zinc chloride is formed, which is added to the main bulk of zinc chloride concentrated from the press filtrate, and the whole eletcrolyzed.

Evaporator.

This consists of a long iron tank fitted with cross partitions and a central partition running the entire length, thus dividing the tank into several compartments. The inside is lined with thick sheet lead.

The tank is set in a producer heated furnace, and is so arranged as to allow the lower pans to be made much hotter than the upper ones, a gradual increase of temperature being produced as the solution travels from the top to the bottom tanks.

The solution enters the topmost divisions in the tank and from them overflows into the next lower down, and proceeds thus until it reaches the lowest tanks. It is gradually concentrated as it travels down and matters are so arranged that it is a frothing mass containing about 90 per cent $ZnCl_2$, when it is in the two deep pans at the end. From these pans it flows into enamelled iron pans more strongly heated, which drive off the last portions of water. The chloride froths considerably at this stage, owing to small quantities being decomposed by the high temperature in the presence of water—HCl and ZnO being formed. This may be reduced to a certain extent by adding strong HCl during the frothing stage.

With careful working the amount of basic material formed is not great. It has been proposed to avoid this decomposition by performing the last stages of evaporation in a vacuum evaporator, but this has not yet been tried. The fused zinc chloride is colorless and appears quite clear through a depth of six inches. When basic it loses to a certain extent its transparency. If colored gray, with traces of organic matter it can be cleared by stirring in a little potassium nitrate.

Preliminary Electrolysis.

In order to decompose slight traces of water and the basic compounds which are formed at the "frothing stage" the chlorides are subjected to a preliminary treatment by electrolysis.

This takes place in a brick-lined, sheet-iron vessel, fitted with a cover from which is suspended cheap carbon blocks which dip into the molten chloride, and form the anode. The cathode is composed of a bath of molten zinc. The chloride is kept molten by the passage of the current, no external heat being required. The preliminary electrolysis continues until all the water and basic compounds are decomposed. The chloride so treated is perfectly neutral and corresponds by analysis to the formula $ZnCl_2$.

The power required for this preliminary work is about 10 per cent of that required to decompose the purified $ZnCl_2$ into zinc and chlorine.

The Electrolysis Vats.

The principle of the vats is shown in Fig. 2. These consist of sheet-iron cases lined with firebrick. The cover is made of cast iron and forms a gas-tight joint with the body

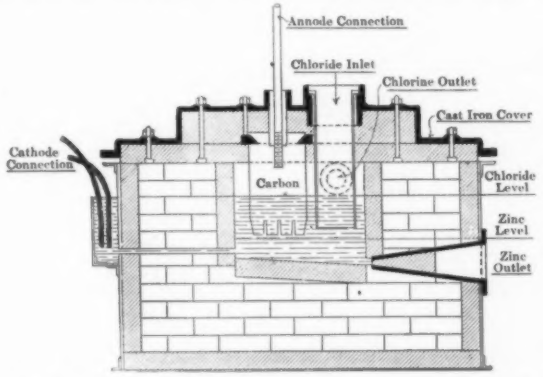


FIG. 2.—CONSTRUCTION OF VATS FOR ELECTROLYSIS OF MOLTEN ZINC CHLORIDE.

of the vat. From the cover are suspended blocks of carbon, which reach nearly to the bottom of the inside of the vat body. An outlet for the chlorine is arranged from the vat cover, or sides.

The cathode is composed of the molten zinc and connection is made from this to the external circuit by means of a small tank fastened to the outer case, and communicating with the interior by means of a small channel cut through the brickwork. The zinc fills this tank and the conductors dip into it, thus making a good connection, the zinc becoming solid. If the chlorides are pure, electrolysis proceeds quite smoothly, molten zinc being run out and fresh chloride run in at regular intervals.

With proper appliances for handling the chlorides, one man can look after about ten vats. The interior of the vat is maintained under a slight suction, so that in the case of a leaky joint air is drawn in and chlorine cannot escape to the outside.

A current density of about 400 amperes per square foot of cathode surface is employed, and the potential difference between the vat terminals is 4.5 volts. The temperature of a vat running normally is 450°. It is the custom to dilute the chloride of zinc with sodium chloride, the internal resistance of the vat and fuming being thus reduced. The chloride of zinc is diluted until the percentage of Zn is 28 per cent.

Up to the present vats of 3,000 amperes have been used, but designs of one to work at 10,000 amperes have been prepared, and it is expected that this will result in the lowering of the potential difference. It has been found possible, with perfectly pure $ZnCl_2$, to obtain a yield of zinc extremely close to the theoretical. In one experiment, where the current was measured by a copper voltameter, a yield of 1.193 grams of Zn per ampere hour was obtained as against a theoretical of 1.2.

Pumping the Chlorine.

All that now remains to complete the cycle is to pump the

chlorine gas from the vat, and put it under a pressure of 25 pounds per square inch for use again in the transformer. Easy as this may appear, it has caused a great deal of trouble and expense to accomplish. One difficulty was that all forms of asbestos or rubber packing were rapidly destroyed by the action of the chlorine, and a system of metallic packing had to be arranged.

Another trouble was caused by the fact that damp chlorine —even if moisture were present in extremely small quantities —caused the destruction of the iron work of the pump. The lubrication of the pump was another great difficulty, since almost all hydrocarbon oils were decomposed by the chlorine, resulting in the valves and other parts of the pump becoming coated with a hard deposit of carbon which stopped the efficient action. All these troubles have been overcome and chlorine has been pumped for weeks on end with no difficulty.

It has been proposed to pump the chlorine by means of a turbine blower, such as is sometimes used in blast furnace work, but this has not been tried.

To supply the waste of chlorine which occurs in the different stages—about 5 per cent per cycle—chlorine has to be supplied from external sources. This is usually obtained from bleach powder. Chlorine so obtained is wet, and in order to extract the moisture it is drawn through a series of lead towers packed with a solid zinc chloride and connected with the suction of the pump. The accumulated moisture gradually causes the zinc chloride to form a strong solution, which falls to the bottom of the tower and makes its exit into a tank from which it is periodically removed, concentrated, fused, and repacked in the towers. The drying of the gas is thus produced at the slight cost of fuel for evaporating and labor for packing.

Conclusion.

This completes the description of the cycle as applied to a material having the composition of Broken Hill ore. If the ore to be treated is more complex, the number of processes which the chlorides pass through is increased.

Thus, if the ore contains copper in addition to the other metals, the treatment would be as follows: The chlorides from the transformer would be dissolved in water and the liquor stirred with copper plates, replacement of the silver by copper then taking place, the metallic silver being deposited on the plates.

Oxidation and precipitation of the iron and manganese and the separation of these oxides together with the lead chloride and gangue by settling or filter process would then proceed in the manner already described. The clear solution containing only copper and zinc chloride would then be stirred with granulated zinc in sufficient quantity to replace all the copper present, the latter metal after allowing the liquid to settle, being obtained in the form of cement copper, while the clear solution containing now only pure zinc chloride, would be evaporated and electrolyzed in the usual manner.

If gold were present it would be precipitated with the silver and recovered with that metal. Antimony, if present, would distill from the transformer with the sulphur, and condense before the substance in the form of the liquid pentachloride from which it can be replaced indirectly by means of zinc.

The author thinks that treatment of arsenical, antimonious, telluride and other ores for gold could, with modifications, be carried out, and thus render workable a good deal of material which at present cannot be treated.

Cheap power is essential for the economical working of a process like this, and therefore the works should be situated near to a cheap source—such as a waterfall; the works should also be arranged to allow the material to flow from one department to the next, in order to avoid handling. This could conveniently be done by either building the works on a hillside, or by placing them in a building which contains as many stories as there are operations—six in the case of Broken Hill ore. The crushed ore would be elevated to the top story, where the converter would be placed, and the fused chloride flow from this through the different stages, ending with the electrolytic vats on the lowest floor.

Faraday Society Meeting.

The tenth ordinary meeting of the Faraday Society was held on December 19, 1904, at the Institution of Electrical Engineers in London. Mr. J. Swinburne, vice-president, occupying the chair.

The chairman announced that the council had made the following nominations of officers and members of council to serve during the coming year. The election will take place at the annual general meeting, to be held in February:

President.—Lord Kelvin, O. M.

Vice-Presidents.—Prof. A. Crum Brown, F. R. S., Sir W. Huggins, P. R. S., Sir Oliver Lodge, F. R. S., Dr. Ludwig Mond, F. R. S., Lord Rayleigh, O. M., Alexander Siemens, Pres. I. E. E., James Swinburne.

Council.—G. T. Beilby, Bertram Blount, W. R. Cooper, Sherard Cowper-Coles, Prof. A. K. Huntington, Dr. R. A. Lehfeldt, W. Murray Morrison, Dr. W. S. Squire, Dr. O. J. Steinhart, Prof. E. Wilson.

Treasurer.—Dr. F. Mollwo Perkin.

The Electric Furnace.

M. Adolphe Minet communicated Part II. of his paper on "The Electric Furnace: its Origin, Transformations and Applications," which was read in abstract by Mr. R. S. Hutton.

In the first and introductory part of his paper, which was read before the Society on June 9 last, M. Minet showed that the historical development of the electric furnace falls into the three following periods:

1. Laboratory furnaces, 1808 to 1886.
2. Industrial furnaces, 1886 to 1890.
3. Development of industrial applications from 1890 to the present day.

The present instalment, after some preliminary general remarks on the e. m. f., current densities, and yields, with arc furnaces and resistance furnaces, respectively, proceeds to deal in detail with the evolution of the furnace during the first period. The subject is sub-divided as follows:

(*a*) *Original Furnaces*, namely, Davy's arc, Pepys's resistance furnace for the cementation of iron, and Davy's and Napier's cathode furnace.

(*b*) *Electrolysis at High Temperatures;* the production of aluminium and its alloys, and the alkali and alkaline earth metals.

(*c*) *Miscellaneous Furnaces*, of which the most important are the early Despretz and Pichon furnaces, Berthelot's electric "egg" for the synthesis of acetylene, Siemens' and Huntington's well-known steel and other furnaces, the Louis Clerc furnace, interesting as the progenitor of the famous Moissan type, and Borchers' resistance furnace.

The paper, which is illustrated with diagrams of the principal furnaces alluded to in the text, closes with a bibliography covering the period dealt with. A final instalment will discuss the furnaces of the second and third periods.

Mr. W. B. Clarke referred to the importance and difficulty of making temperature determinations of electric furnace processes, and expressed the hope that M. Minet would deal with that question in Part III. of his paper.

Mr. L. Gasta handed round some "siloxicon" and described its properties.

Mr. E. Kilburn Scott described some of his experiences in the early days of the manufacture of calcium carbide.

The chairman described a novel type of furnace that he had once hit upon by accident, and which he thought might turn out to be useful. He was sending a current through fused

barium chloride, and the temperature having been allowed to get too high the cast-iron crucible melted and formed a fused mass that was kept together by an outer skin of solidified chloride. This suggested the use of a neutral substance like barium chloride for heating to a very high temperature alloys or metals which it was desired not to contaminate with impurities.

ELECTROLYTIC ANALYSIS OF COBALT AND NICKEL.

A paper on this subject, by Dr. F. MOLLWO PERKIN and Mr. W. C. PREBBLE, was then read by Dr. Perkin.

Cobalt.—The aim of the experiments was to obtain bright deposits of the metal that should be quantitatively accurate. Various reducing agents were first of all employed, but only with sodium hypophosphite were brilliant deposits obtained, and then phosphorus was found to be present. Salts of the various organic acids were next tried, but only ammonium oxalate and tartrate gave satisfactory results, the latter having the better appearance. Traces of carbon, however, were deposited with the metal. Ammoniacal solutions of ammonium borate were also tried. The most satisfactory results were obtained with a solution containing an alkali phosphate and a little phosphoric acid; not only is the deposit bright, but the analytical numbers are also good. By this method the whole of the metal can be thrown out in from 3 to 3½ hours.

In carrying out this process it was found that only a small quantity of ammonium or sodium phosphate may be added; otherwise, the double salt of cobalt and the alkali phosphate is precipitated, and when once precipitated it can only be redissolved again with great difficulty. The following method of procedure gave the most satisfactory results: About 1 gram of the cobalt salt was dissolved in about 50 c.c. of distilled water, and then 5 c. c. of a 5 per cent solution of phosphoric acid added. To this solution 25 c. c. of a 10 per cent solution of ammonium dihydrogen phosphate [$(NH_4)H_2PO_4$], or better, sodium dihydrogen phosphate (NaH_2PO_4) was added; the solution was then made up to 130 c. c. and electrolyzed, a gauze flag electrode (ELECTROCHEMICAL INDUSTRY, Vol. I., page 414) being employed as cathode. After the electrolysis has proceeded for half an hour or so there is usually a brown deposit of cobalt oxide on the anode. In order to dissolve this deposit off, about 0.5 gram of hydroxylamine sulphate or chloride is added to the solution, when the deposit almost immediately vanishes. Very often 0.1 gram of the hydroxylamine salt is quite sufficient. The addition of a few c.c. of formaldehyde solution has the same effect, but the action is much slower. The anode deposit is much more marked when hot solutions are electrolyzed, but there is no difficulty in removing it, as already described. If the solution is heated to too high a temperature, there is a tendency for the double salt of sodium and cobalt phosphate to be precipitated out. The phosphoric acid is added in the first place to prevent the precipitation of this double salt, because if it is once precipitated a considerable quantity of phosphoric acid must be added in order to dissolve it again. But when there is too much phosphoric acid in the solution the deposition of the cobalt is extremely slow. It is, in fact, always an advantage, after the bulk of the cobalt has been deposited, to add a few drops of very dilute ammonium hydrate to neutralize the excess of acid liberated by the electrolysis.

Nickel.—Similar solutions were tried for nickel deposition. In this case splendid results were obtained with the borate solution, while the phosphate solution, which gave such good figures in the case of cobalt, was not at all satisfactory. Using solutions containing ammonium borate and ammonia for nickel, the analytical results, from a quantitative point of view, are magnificent, and although the deposited metal has not the brilliant burnished appearance often obtained by other methods, it is not unsightly, having a matte, silver-greyish appearance.

The authors have not yet succeeded in devising an accurate electrolytic method for the separation of nickel of cobalt, and they find that the methods given in books are unreliable.

In the discussion which followed, Dr. O. J. Steinhart discussed the problem of the separation of the metals. The present method is based on the fact that cobalt is more easily oxidized by chlorine than is nickel. He also referred to the possible industrial bearing of the work of the authors in connection with nickel extraction.

Dr. L. Dobbin, in a written communication, drew attention to the work of Marshall on the same subject. In the case of cobalt he merely added a little nitrate to the ordinary double sulphate solution.

Dr. Perkin said that the presence of nitrate rendered deposition very slow.

TIN.

Mr. F. GELSTHARP presented two short papers on the electrolytic preparation of tin paste and a note on the electrolytic recovery of tin, respectively, which were read by the secretary. These two papers are printed in full on other pages of our present issue.

In the discusion which followed, Mr. W. C. Prebble thought an endless band would be a less crude method of removing the paste than that given by the author. Regarding the presence of arsenic, which was hurtful here, it was curious that in the case of zinc this actually causes a spongy deposit to form.

Dr. O. J. Steinhart gave a few particulars of an electrolytic detinning process that was now successfully treating in London seven or eight thousand tons of scrap per annum. A great difficulty found in practice was to dry the tin sponge without its firing.

Dr. T. C. Cain did not think it probable that sodium was deposited with the tin.

Mr. H. Sayer asked what percentage of the tin was recovered. An objection to an alkali process was that it would not remove the tin from the alloy at the surface junction of the two metals.

Mr. A. C. Bawtree sent in a written communication describing an electrolytic method for making tin powder that he had worked out some years ago. His solution contained 0.5 per cent $SnCl_2$ and 0.33 per cent $SnCl_4$, and was worked cold with an e. m .f. of 2.32 volts. This solution successfully overcame the difficulty of crystalline deposits.

Mr. H. S. Coleman also communicated some of his own experiences in tin deposition. He had worked under conditions similar to those described by Mr. Gelstharp. He found, however, that if the bath were worked between 50° and 60° C., no diffculty was experienced in starting the current, and the scrap was stripped clean in twenty minutes.

Improvements in the Treatment of Slimes for Winning Gold.

The interesting article by Messrs. Clement Dixon and M. Torrente, published in our June issue (Vol. II., page 215), contained a concise review how the problem of treating slimes for winning the gold contained in them came up, and was solved in the South African gold fields. This article gave a full review of the development and present practice of gold metallurgy on the Rand.

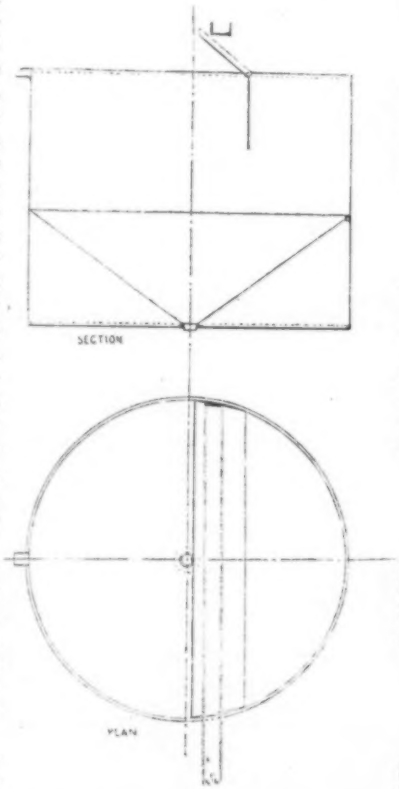
In a recent paper presented before the Chemical, Metallurgical and Mining Society of South Africa, and published in the August number of the journal of the Society, Mr. M. TORRENTE, who, as the successor of Messrs. Charles Butters and von Gernet, has been most intimately connected with the developmnet of the electrolytic precipitation process on the Rand, brings forward what he considers a very decided improvement in the treatment of slimes. In the following we reproduce this paper practically in full.

Although the paper only refers to the settlement and separation of the slimes from their solutions, the fact that a good continuous settlement can be obtained, and perfectly

clear solutions continuously withdrawn (without in any way affecting the value of the residues at present obtained) may, in itself, be considered a step further in the right direction. In addition, some other very considerable advantages may be derived by following this procedure.

In his experience of slimes treatment, Mr. Torrente has arrived at the conclusion that the problem of a continuous treatment presents the greatest difficulties. These may not be so considerable in treating the fresh slimes as they come from the battery, but when dealing with old accumulations where the dissolving of the gold may take anything from twelve to twenty-four hours, the obstacles in the way are evident to anybody experienced in these processes. The first man who experimented on these lines was Mr. Ernest T. Rand, formerly manager of the Simmer and Jack cyanide works. To a certain extent his results were satisfactory, but he had to contend with the same disadvantage which other processes have shown since, viz, that a considerable amount of slimes went out of the plant untreated.

After some experimenting, Mr. Torrent gave up the idea of continuous treatment. The drawbacks of a decantation slimes plant are many, and the margin for improvements still very great. The greatest drawback, in the author's opinion, is the time a plant has to remain idle while decantation is going on. To be able to work continuous settlement on a plant means immediately doubling at least its capacity in one case, and halving its initial cost in the other. By his new method Mr. Torrente considers that he has fairly well attained these two objects.



APPARATUS FOR SLIMES TREATMENT.

In the first instance the author mentions that he does not interfere with the treatment of the slimes, which may be carried out as heretofore.

Having first ascertained that the gold is dissolved in the slimes charge, the charge is slowly pumped to the separator, which consists of a tank with a sharp, conical bottom and a vertical partition going down to about two-thirds of its depth or near the level of the inverted cone's base. This partition, which may or may not exceed the height of the liquid in the tank, is continued in the form of an inclined plane for some distance above the level of said liquid. The tank, before it starts work, is filled with water or solution up to the solution outlet, and the slimes pulp is delivered on the inclined plane, where it spreads over the whole of the surface in a very thin layer. The pulp thus enters the solution without the slightest splash, and is bound to continue its downward movement on the one side of the partition. The quantity of pulp, in proportion to the mass of solution contained in the settler, is so small that it becomes diluted to an extent which is practically impossible in any other way. The fine particles of slimes assert to great advantage their specific gravity and continue in their downward movement after they have arrived at the lower edge of the partition, accumulating in the cone of the vat, from the bottom of which they are discharged. The solution, once freed from the slimes it originally contained, having now less gravity, rises in the contrary direction to the column of pulp on the other side of the partition and is discharged through an outlet provided for this purpose near the top of the tank.

The experiments so far made have given the greatest satisfaction. The following tables showing comparative results, will prove interesting. In every case, the pulp taken has come from the slimes plant after the gold has been dissolved, as far as the slimes in question have permitted, during a treatment of twenty-four hours. The washes have also been given as much as possible under working conditions. It must be well kept in mind that the slimes in question are old accumulated slimes, very difficult to treat. Therefore, most figures will be found very high compared with the actual work now done with fresh slimes taken direct from the batteries.

Experiments in Continuous Settlement.

The slimes used for this experiment were taken from the company's dam nearest to the spruit and can be considered as the worst possible for cyanide treatment, being highly contaminated with acid and organic matter, the time necessary for dissolving the gold they contain being eighteen to twenty-fours hours.

Quantity of dry slimes taken, 200 pounds.
Capacity of settler, 800 pounds.

First Wash.—Time taken for settlement, *i. e.*, getting the charge through, 8½ hours.

Ratio of slimes to solution	4:1
Original assay value of the slimes	4.5 dwts.
	Assays.
Solution added before treatment	13 grs.
Solution from pulp after treatment	1 dwt. 12 grs.
Value of undissolved gold in slimes	15 grs.
Value of the residue (59.3 per cent moisture), including gold carried in moisture—	
By calculation	2 dwts. 16 grs
By assay	1 dwt. 12 grs

Second Wash.—Time taken, 10½ hours.

Solution used for washing	1 dwt. 3 grs.
Pulp solution	1 dwt. 7 grs.
Value of undissolved gold in slimes	15 grs.
Value of the residue (56 per cent moisture), including gold carried in moisture—	
By calculation	2 dwts. 3 grs
By assay	1 dwt.

Third Wash.—Time taken, 12 hours. Assays.

Solution used for washing	4 grs.
Pulp solution	13 grs.
Value of undissolved gold in the residue	12 grs.
Value of residue (49.2 per cent moisture), including gold carried in moisture—	
By calculation	21 grs.
By assay	1 dwt.

This experiment compares with the treatment of slimes as now carried out as follows:

Same Kind of Slimes (After Gold Has Been Dissolved).

First Wash.—Time taken for settlement, 12 hours.

Ratio of solution to slimes	4:1
Original assay value of slimes	4.5 dwts.
	Assays.
Value of solution before treatment	13 grs.
Value of solution after treatment	1 dwt. 12 grs.
Value of undissolved gold in slimes	12 grs.

Solution decanted 1½ tons to 1 ton of slimes.
Value of residue, including the gold still carried in the remaining solution ..... 4 dwts. 6 grs.

Second Wash.—Time taken for settlement, 12 hours.
Ratio of solution to slimes..... 4:1
Value of solution used for washing..... 4 grs.
Value of solution after pulping..... 1 dwt.
Solution decanted 1½ tons to 1 ton of slimes.
Value of residue, including the gold still carried in the remaining solution ..... 3 dwts.

Third Wash.—Time taken for settlement, 60 hours.
Ratio of solution to slimes..... 4:1
Value of solution used for washing ..... 4 grs.
Value of solution after pulping..... 13 grs.
Value of residue (discharged with 50 per cent moisture), including the gold carried in the moisture... 1 dwt.

These results apply solely to charges like the above, which has been treated under the most favorable conditions.

SECOND EXPERIMENT.

The slimes used in this experiment were taken from the dam near the cyanide works. In the ordinary way they are most difficult to settle, although more amenable to cyanide treatment.

Quantity of dry slimes taken, 200 pounds.

First Wash.—Time taken to put the charge through, 24 hours.
Ratio of slimes to solution..... 1:8.4
Original assay value of the slimes ..... 3.8 dwts.

Assays.
Solution added before treatment..... 14 grs.
Solution from pulp after treatment ..... 1 dwt.
Value of undissolved gold in slimes..... 12 grs.
Value of the residue (61.1 per cent moisture), including gold carried in moisture—
By calculation ..... 2 dwts. 1 gr.
By assay ..... 1 dwt. 12 grs.

Second Wash.—Time taken, 12 hours.

Assays.
Solution used for washing ..... 10 grs.
Pulp solution ..... 22 grs.
Value of undissolved gold in slimes..... 7 grs.
Value of the residue (58.5 per cent moisture), including gold carried in moisture—
By calculation ..... 1 dwt. 5 grs.
By assay ..... 19 grs.

Third Wash.—Time taken, 11 hours.
Solution used for washing ..... 4 grs.
Pulp solution ..... 10 grs.
Value of undissolved gold in slimes..... 7 grs.
Value of the residue (55.9 per cent moisture), including gold carried in moisture—
By calculation ..... 18 grs.
By assay ..... 15 grs.

The first wash was put through in twenty-four hours and with about double the quantity of solution for experimental purposes only. The time taken for the second wash applies to the first for purposes of calculation.

The second experiment compares with the treatment of the same class of slimes as now carried on as follows:

First Wash.—Time taken for settlement, 24 hours.
Ratio of solution to slimes..... 2:6.1
Original assay value of slimes ..... 3.8 dwts.

Assays.
Value of solution before treatment ..... 23 grs.
Value of solution after treatment ..... 2 dwts. 2 grs.
Value of undissolved gold in slimes..... 22 grs.
Solution decanted 1.6 tons to 1 ton of slimes.
Value of residue, including the gold still carried in the remaining solution ..... 2 dwts. 21 grs.

Second Wash.—Time taken for settlement, 60 hours.
Ratio of solution to slimes, 3 tons to 1 of slimes.
Value of solution used for washing..... 4 grs.
Value of solution after pulping ..... 18 grs.
Solution decanted 2 tons to 1 of slimes.
Value of residues, including the gold still carried in the remaining solution ..... 1 dwt. 4 grs.

Owing to their nature, the treatment of these slimes had to be curtailed to two washes instead of the former three, the time not being sufficient to allow three settlements.

By following the process above described a slimes plant of almost any size can now be built with two or three treatment vats and three separators. The experiments have clearly shown that slimes to the amount of two-fifths of the total capacity of any given tank can be separated in twenty-four hours, and the return of the pulp from one separator to another with the solutions as now employed constitutes a very thorough washing.

The procedure would be as follows: The treated pulp is pumped to the first separator; the overflowing solution, which constitutes the strong solution, has its gold precipitated in the zinc boxes. From the bottom of the first separator the slimes are pumped to the second separator after the addition of the necessary amount of intermediate solution to make them into a thin pulp. The same operation is repeated in the third separator, substituting precipitated solution or water, as may be most convenient for thinning the pulp. From the bottom of the third separator the slimes flow by gravity into the residue dam, or are pumped there, as may be found most convenient. As every separator constitutes a wash, the number of these will be determined by the value of the slimes and their nature. In some plants two may be found to be sufficient; in others it may be advantageous to put three or even four. So far the author has found that three have done very satisfactory work.

How far the capacity and its initial cost is affected is evident. The author thinks he may safely say that the capacity of any of the existing plants can be doubled at once at a very small cost. In new ventures a plant consisting of three 400-ton treatment vats and three 500-ton separators will easily handle 200 tons of dry slimes per day, that is to say, that for about one-half the present capital expenditure a plant can be erected to treat double the amount of slimes that a plant would treat if constructed on existing lines. The experiments were all carried out at the Crown Reef.

Notes on Electrochemistry and Electrometallurgy in Great Britain.

(From Our Special Correspondent.)

THE FARADAY SOCIETY MEETING.

The fact that four papers were down for reading at the December meeting of the Faraday Society, on December 19, led those attending to expect a protracted meeting. The attendance was rather smaller than usual, owing to the proximity of the holidays. Mr. Swinburne presided. The first paper consisted in the second instalment of M. Adolphe Minet's serial on "The Electric Furnace, Its Origin, Transformations and Applications." The first instalment was read on June 9, and a rather scathing communication sent to the *proceedings* by Messrs. Hutton and Morrison. It was therefore somewhat surprising to find Dr. Hutton generously acting as *locum tenens* for M. Minet. A brief summary of the paper was given, and reference was made to the work carried out by Prof. Richards, of Lehigh University as to what actually constituted the economic yield of a furnace. M. Minet's encomium on Borchers was mentioned as a useful confession which probably described Borchers' true relation to electrochemistry. The reference in question was worded as follows: "Like most others who have plenty of ideas, Borchers

only attached secondary importance to some of his laboratory results. Then later on other investigators made the conceptions of the German investigator the basis of their researches, without troubling to indicate the source. As a result, Borchers showed a tendency to restrain his appreciation of the work of others, whether they had worked on different lines or had forgotten to do him justice. But time, happily, allows all things to be seen in their right perspective."

The resulting discussion was strikingly irrelevant. One member commenced to read a small paper on Acheson's new product "siloxicon," another spoke of temperature determinations. Mr. Swinburne's remarks were practically confined to an anecdote concerned with a laboratory blunder when some barium chloride was being melted. The property of being intensely heated, sufficiently indeed to reduce iron without being oxidized, suggested that barium chloride might possess valuable properties in electro-thermic work. A vote of thanks was, of course, accorded to M. Minet, whose history of the electric furnace will probably run into a third or fourth paper. It would surely be best not to discuss any of these, but to have a sort of field day at the conclusion and discuss the contributions as a whole.

Dr. Perkin next introduced the paper by himself and Mr. Prebble on the "Electrolytic Analysis of Cobalt and Nickel." The introduction was slightly disconnected, not following the same order in the treatment of the subject as the paper. Some rather interesting samples of deposited zinc and nickel were shown. The discussion was confined to the reading of a communication of Dr. Dobbin, and to some remarks by Dr Steinhart. In the former, reference was made to a paper by Marshall, read before the British Association in 1898, in which the addition of traces of nitrate was suggested. A peculiar rise of voltage on the exhaustion of the nickel from the solution was then recorded. Dr. Steinhart dealt chiefly with the commercial side of the question. The difficulty of separation was chiefly due to both metals forming peroxides. Dr. Perkins' success was probably due to the fact that the salts used had a reducing action, limiting the formation of peroxide. In reply to Dr. Dobbin, Dr. Perkin pointed out that the addition of traces of nitrate greatly increased the time needed in any such operation. He had not noticed the rise in voltage cited.

The next two papers by Mr. F. Gelstharp: the first a note on the "Electrolytic Preparation of Tin Paste," and the second a note on the "Electrolytic Recovery of Tin Scrap," were discussed together. In regard to the first, Mr. Prebble suggested the use of an endless band to carry the tin plate outside the cell, in place of the scraper used. The question of the presence of arsenic and difficulties of drying were cited. Dr. Steinhart then intervened, and for the rest of the evening championed the absent author of the paper. Tin recovery processes in England chiefly suffered by reason of the paucity of tin scrap. He endorsed the author's remarks on agitation. The chief difficulties were those of subsequent utilization; the oxidation was terribly bad and smelting difficult. Only a low voltage was needed, and the current efficiency was very high. Indeed, in a case brought under his notice, higher than that allowed for by Faraday's law. Dr. Perkin here intervened to suggest that extraordinary efficiencies were due to iron and copper impurities. No one countered Dr. Steinhart's remark that the current efficiency figures were those of several months by questioning the calibration of the meters. Mr. Swinburne said a few works of his own experiences as an investigator, which were chiefly concerned with the difficulty of getting tin scrap. Mr. Sayer then asked what was the actual percentage of tin recovered? Only from 2¼ to 5 pounds of tin were used per box of plates (weighing 100 pounds); figures claiming a recovery of 3 per cent to 9 per cent were surely inaccurate. Dr. Steinhart replied that the recovery efficiency was 95 per cent, and, replying to a further query (as to whether the tin left on the scrap was deleterious in iron and steel manufactures), remarked that the iron dust was briquetted and found a ready sale among smelters who insisted on accuracy within a second or third place of decimals in the contents. The meeting then adjourned.

The Germicidal Action of Copper.

Electrical engineers who have followed the recent experiments of the Washington Department of Agriculture as to the value of copper sulphate in the prevention of the growth of algae and certain pathogenic organisms present in water supplies, will, unless interested financially in copper refining, forsee without much enthusiasm another factor which may make for dearness in the future. A perusal of a recent paper by Dr. Rideal and Dr. Baines, read before the last Congress of the Sanitary Institute at Glasgow, can only tend to a strengthening of these forebodings. Dr. Rideal and his colleague show, firstly, that, comparing the action of copper sulphate and copper chloride on water inoculated with the bacillus coli and bacillus typhosus, the bactericidal efficiency is dependent on the absolute amount of ionic copper present, that copper foil having a surface of 100 square centimeters introduced into a 100 c. c. flask containing polluted water, effected a complete sterilization within twenty-four hours. In the paper read no hypothesis is put forward to account for this action. Your correspondent having talked the matter over with Dr. Rideal is not yet prepared to advance any theories at present. He can, however, state that Dr. Rideal's further experiments in this direction will probably be along the lines of pure electrochemistry rather than entirely chemical—if the action be chemical—as heretofore.

The Aluminium Industry in Great Britain.

The British Aluminium Company is about to greatly increase its capital, and the chairman, Mr. J. D. Bonner, stated at a recent meeting that not less than £490,000 additional capital would be needed. Of this, part is to be raised by a debenture issue, and part by the issue of shares. The Loch Leven water-power scheme is about to be undertaken, and the works at Larne, Greenock and Milton extended. It is hoped that the Loch Leven scheme will be finished by the end of 1906, in order to meet the market requirements which it is then anticipated will be considerable.

Metallurgical Papers Read in December.

The meeting of the Institution of Mining and Metallurgy, held on December 15, had before it four papers. The first, by Messrs. R. A. Thomas, and W. P. O. Macqueen, was concerned with "The dust in the air, and gases from explosives in Dolcoath Mine (Cornwall), and the efficacy of methods of dealing with them." For limiting the amount of dust in the air the authors have found a coarse spray most effective in machine drilling. In blasting, in order to clear the air of dust, a water blast of finely divided water and air, discharged immediately after the blast is extremely effective. The same device has also been found to immediately reduce the percentage of carbon monoxide in the air after an explosion from one-third to one-fifth of its volume. In regard to carbonic acid gas, continuous ventilation seems the only remedy.

The second paper, by Mr. E. Harris, dealt with the "Permanganate Chlorination Process at Bethanga" (New South Wales). The solvent employed is a modification of the Etard solution and consists of 110 pounds of commercial sulphuric acid, 90 pounds of common salt, and 12 pounds of potassium per 1000 gallons of water. These figures are more or less experimental, as the present process has only lately superseded the chloride of lime method. The advantages claimed for it over the older methods are (1) the less amount of free chlorine with consequent easier work to the men; (2) quicker extraction and less clogging of the filter beds; (3) the ready means of gauging the solvent strength afforded by the deep ruby color of the working solution; and (4) the obtaining of a cleaner precipitate with easier smelting.

Of the remaining papers, one was descriptive of the methods of mining carried on in the St. David's Mine in North Wales, and the other describing a new slag car.

The Institution of Mechanical Engineers met on the following evening, December 16th, to conclude their discussion on the "Impact Tests on the Wrought Steels of Commerce" (to which reference was made in my last letter) and to receive a new paper by Captain Sankey and Mr. J. Kent-Smith on "Heat Treatment Experiments with Chrome-Vanadium Steel." The paper is a long one, occupying some 48 pages of printed matter, illustrated with eleven plates of micro-photographs and diagrams of tests. Of the steels tested by the authors the best was that made from an 18-ton open-hearth furnace, and numbered 349. It contained 0.297 per cent carbon, 0.059 per cent silicon, 0.394 per cent manganese, 1.066 per cent chromium and 0.169 per cent silicon. Its tensile strengths, when raw from the rolls, showed a yield point of 369 tons, and an ultimate stress of 54.1 tons per square inch. Its elongation on 2-inch gauge was 24.0 per cent. On submission to impact tests, the figure for the average foot-pounds absorbed was 3.1, the test pieces ⅜ inch broad, the depth of the notch being such that 0.137 inch of metal remained. When tested on Professor Arnold's alternation testing machine, a test piece 4 inches long and ⅜ inch square withstood 1906 alterations before breaking down. The motion of the free end being 9-16 inch on either side of the center line.

The heat treatment was carried out in a muffle of the electric resistance type, consisting primarily of an earthenware tube around which was spirally wrapped thin platinum foil. The current was at 100 volts pressure, 27 amperes being required when the temperature of the muffle was kept at 1200° C. and 1250° C.

Except as regards such treatment as ruins the steel the chrome vanadium steel is distinctly less sensitive to heat treatment than is a pure carbon steel, while the impact figure is considerably larger. A great dissimilarity exists between annealed chrome vanadium steel, and the best oil-tempered carbon steel (containing 0.468 carbon). In the case of the annealed chrome vanadium steel the characteristics are maintained throughout the mass, whereas in the case of an oil-tempered steel in large masses the characteristics depend entirely on the depth to which the effect penetrates. The oil-tempering of chrome vanadium steel, however, further improves its quality in regard to elastic limit and impact test; on the other hand the alternations (Arnold's test) are diminished. When water-quenched, chrome vanadium steel agrees more closely with 0.254 carbon steel.

The diminished sensitiveness to heat treatment is a matter of the greatest practical value. It should be noted, however, that if the tensile strength is to be maintained, the annealing temperature should not exceed 650° C., while if the best all-round result is required, the annealing temperature should be 900° C.

Rather over six months ago a leading article in *The Engineer*, entitled "Ideal Steel," attracted considerable attention. It almost seems as though Captain Sankey and Mr. Kent-Smith have succeeded in producing a steel which answers to the ideal of *The Engineer*. Who, however, will be rash enough to say that further progress is unattainable, because so much has been achieved?

MARKET QUOTATIONS.

Chemicals have been remarkably steady during the month of December, the quotations for December 28 agreeing very closely with those for November 30. Caustic soda, copper sulphate, and bleaching powder are unchanged. Shellac has, however, fallen from 240s. to 200s. per cwt.

Fine Para rubber is still firm; there was a slight fall in the middle of the month to 4s. 10d.-5s. 0d., recovering to 5s. 3d. Meanwhile, small lots from Ceylon are fetching 5s. 10d. to 6s. 0d. per pound. Gutta percha is quoted at 8s. 0d.

Platinum still remains 81s. 6d. per ounce. Aluminium is stationary, and electrolytic copper bars rod and sheet are unchanged. Copper refined by the wet process weakened a little in the middle of the month, since recovering. Cleveland pig iron has risen further to 51s. 0d. Block tin, after declining to £133 per ton, has risen to £134-£135, or £3 less than at the end of November. Zinc sheets are stationary at £29.7.6 per ton. Brass sheets and tubes are practically unchanged.

London, January 4, 1905.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

BY GEORGE P. SCHOLL, PH. D.

ELECTRIC FURNACES AND FURNACE PRODUCTS.

Manipulation of Refractory Material.—E. Thomson, Swampscott, Mass. Assignor to General Electric Co. Patent 778,286, December 27, 1904. Application filed August 28, 1902.

The invention relates to the manufacture of articles of various shapes, such as tubes, plates, etc., from a very refractory material, such as fused silica, in a furnace, with metallic terminals at opposite ends of the furnace. When it is desired to produce a tube, a carbon rod is introduced between the terminals, and is embedded in a charge of granulated quartz or sand. A very strong current is then passed through the rod, in order to melt those portions of the charge contiguous to it, the length of the time during which the current is passed determining the depth to which the fusion takes place, and consequently the thickness of the tube walls. It is stated that the fused coating of quartz around the carbon rod generally does not adhere to the rod, so that the latter can be removed, thus leaving a tube of quartz. If a curved instead of a straight tube is desired, the heating rod is made to curve correspondingly. In such case the heating rod, if it cannot be conveniently withdrawn, is burned out by heating the whole to a high temperature. When the tube to be produced is of relatively large diameter, the heating rod is made hollow. In this case it is provided with a number of fine perforations to allow for the escape of any gases generated during the process. These perforations are so fine that the granulated quartz of the charge cannot pass through them, but they prevent the possible injury to the outline of the tube due to the action of these gases. The exterior of these tubes, which is more or less granular according to the fineness of the raw material used, may be smoothed down by progressively heating it in the electric arc from end to end. The method for the production of a plate or a slab of fused quartz consists in the heating of a perforated plate of carbon, upon the face of which a layer of granulated quartz has been placed, by a heavy current. After the mass has cooled, the plate of fused quartz may be removed, and its surface smoothed off by the electric arc. A series of devices are described in the specification for the production of more complicated structures.

Production of Metallic Magnesium.—T. L. Roberts, New York. Patent 778,270, December 27, 1904. Application filed May 31, 1904.

The process is carried out in a vessel shown in cross-section in Fig. 1. It consists of an iron pot *A*, which is set into a furnace *B* of refractory material and heated from the outside by

means of a burner *C*. It is covered with a slab of slate *D*, provided with an opening *E* through which the anode rod *G* of carbon is introduced. The vessel itself constitutes the cathode. An aperture *H* in the lid serves for feeding the electrolyte, while a tube *N* is provided for carrying off the gases generated during electrolysis. A draw-off tube *O* is used to remove the molten metal, which accumulates on the surface of the bath. During this operation, of course, the heating arrangement has to be withdrawn. Another draw-off cock *J*, at the bottom of the vessel, serves for the withdrawal of the contents of the pot; the cock may be used for withdrawal of the metal, if the electrolyte is of such low specific gravity that the metal will sink to the bottom. A ring *K*, of non-conducting material, preferably porcelain, is used to keep the magnesium, which rises along the sides of the vessel, from coming into contact with the anode. The anode should fit tightly, as the less air is admitted to the pot the better it is, the air seeming to accelerate or assist volatilization more than does an atmosphere of the electrolytic gases. The main point of importance about the process is the composition of the electrolyte, it being claimed that the addition of fluoride of magnesium to the bath will result in the recovery of the metal with little or no loss of the salts of volatilization. The composition of the bath is preferably as follows: Lithium fluoride 120 parts, potassium chloride 80 parts, sodium chloride 40 parts, ammonium chloride 20 parts, magnesium chloride 10 parts, magnesium fluoride 80 parts, and magnesium oxide or carbonate 20 parts. The mixture becomes liquid at a dull red heat, and is then electrolyzed, chlorine and oxygen being formed at the anode. It is stated that the above mixture may be varied considerably and good results still be obtained; certain of the constituents may be omitted or others substituted, or the proportions carried, so long as magnesium fluoride and a chloride of the alkali group are used. A certain amount of lithium salts is advantageous in lowering the specific gravity of the bath, so that the liberated magnesium sinks readily to the bottom, while it also lowers the fusion point of the electrolyte. The ammonium and magnesium chlorides also lower the specific gravity and fusing point of the bath, but as they are very volatile, only limited amounts should be used. The oxide or carbonate of magnesium is not essential, but if added in such amounts that they will be dissolved or suspended in the electrolyte, more or less of it will be reduced along with the fluoride and chloride. They, however, raise the specific gravity of the bath, a fact which has to be taken account of if the metal is to be drawn off from the bottom of the apparatus. No data concerning voltage or current strength are given.

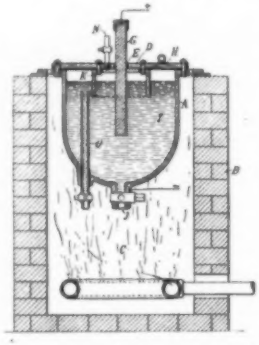


FIG. 1.—APPARATUS FOR PRODUCTION OF METALLIC MAGNESIUM.

Electrolytic Production of Metals and Compounds.

Electrode.—F. J. Briggs, Everett, Mass. Patent 776,490, December 6, 1904. Application filed September 19, 1903.

The electrode is intended to be used in apparatus for the decomposition of the alkaline chlorides, particularly in the inventor's apparatus described in Electrochemical Industry, Vol. I., page 392. The construction provides for an efficient electrical connection, to prevent loss of current as much as possible, while at the same time preserving the feature of easy accessibility for cleaning and adjusting purposes. A block consisting of a number of electrodes is surrounded by a clamp of angle iron, upon the horizontal flange of which it rests. The individual carbons are provided with elliptical tops and arranged in pairs, the tops of each pair being connected by a ribbon of copper, which can be tightened by bolts extending through washers into threaded copper lugs. The latter are integral with a copper plate resting on top of all the electrodes in the block. This plate is connected with the cable from the dynamo.

Process of Producing Salts of Hydrosulphurus Acid.—A. Frank, Charlottenburg, Germany. Patent 777,669, December 20, 1904. Application filed March 8, 1900.

The process aims at producing the hydrosulphites of alkaline earth or earth metals by electrolyzing a practically neutral solution of the bisulphite salt of the metal, the hydrosulphite of which is desired. Electrolysis is carried out in a rectangular vessel which serves as the anode compartment, a series of porous cells, which rest on the bottom of the vessel, constituting the cathode compartments. The solution in the anode compartment consists of dilute sulphuric acid or a suitable salt solution, while the cathodic electrolyte is the bisulphite salt solution mentioned above. The reaction taking place during electrolysis, when calciumhydrosulphite is to be produced, is stated to be as follows: $CaH_2S_2O_6 + 2H = CaS_2O_4 + 2H_2O$. The output of salt with respect to the quantity of current is stated to be the greater, the less acid is contained in the cathode liquid during electrolysis, it having proved advantageous to employ a solution of bisulphite of calcium, which, if possible, does not contain more sulphurous acid than corresponds to the formula $CaSO_3 + H_2SO_3$. The hydrosulphite of calcium, CaS_2O_4, is stated to deposit at the cathode in the form of a heavy crystalline mass, which falls down to the bottom of the cathode receptacle. In this way a decomposition of the salt, either by electrolysis or by the influence of the atmosphere, is avoided. The cathodic electrolyte circulates through the cathode compartments, flowing from a reservoir to the first one and successively through the others, being returned from the last one of the series to the reservoir by means of a pump. Salts of barium, strontium, magnesium and aluminium are stated to be produced in a similar way. No particulars are given concerning the current density or the voltage required for the process.

Process of Reducing Lead Ores.—P. G. Salom, Philadelphia. Patent 778,901, January 3, 1905. Application filed April 17, 1903.

The process, though claimed to be applicable to all lead ores, including argentiferous lead, is described as applied to

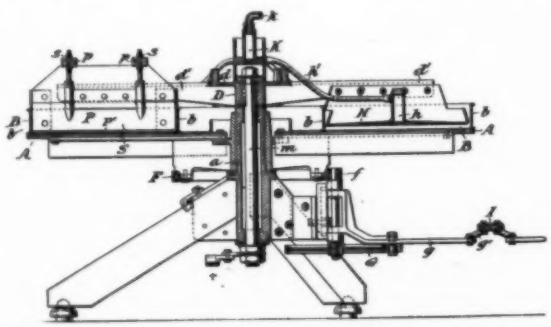


FIG. 2.—APPARATUS FOR CATHODIC REDUCTION OF LEAD FROM GALENA.

the reduction of galena, presumably as carried out in Mr. Salom's works at Niagara Falls, which have been described repeatedly in Electrochemical Industry. The finely ground ore is spread continuously on the cathode in a very thin layer, and is left there until it is completely reduced, the reduced lead being also removed continuously. The apparatus in which the reduction is carried out is shown in cross-section in Fig. 2. Its construction varies to some extent from that described and illustrated in Electrochemical Industry, Vol.

I., page 392, especially as regards the movable part, the anode being movable in the former construction, while the cathode was stationary, whereas in the one under consideration the cathode is movable and the anode stationary. The apparatus consists of a rotating table *A*, which is supported by a fixed central port *a* and carries concentrically an annular electrolytic cell *B*, with inner and outer peripheral walls *b b*. The cell is composed of antimonial lead, and is made the cathode by electrical connections through the central post, which, as shown, is insulated from the floor on which it rests. Movement is imparted to the table by a gear wheel *F*, which meshes with a pinion *f* sitting on the same shaft as the ratchet wheel *G*. The latter is moved by a pawling lever *g*, which is slowly oscillated by a bar *T*, through link connection *g*. The bar moves to and fro, and a whole series of apparatus may thus be kept in motion. The post *a*, around which the table swings, carries above the table a spider frame *D*, insulated from the post and consisting of a central disc *d* and a number of radially-extending arms *d' d'*. The spider frame supports, within the electrolytic cell, an anode in the form of an annulus, which, however, does not form a complete ring, but lacks one segment between two of the arms of the spider frame. The annulus is made up of a series of segmental plates *H*, of antimonial lead, each plate bridging a space between the arms of the spider frame, and having its under surface slightly dished upwards towards a central aperture *h*. This opening leads to a short pipe, which communicates by a rubber tube *h'* with a central chamber *K*, and serves the purpose of conducting away the gases generated during electrolysis. The entire series of anode plates is constituted a single anode by connection to the spider frame by means of the central insulated spindle *m*. The ore is spread upon the rotating table by a supply box *P*, supported from two bars *p p*, which bridge the opening between those two arms of the spider frame, where the segmental plate is lacking, all connections between the box and the spider frame being insulated. The side wall of the box *P*, away from which the cathode rotates, does not reach quite to the bottom of the cell, thus leaving an adjustable slot *p'* for feeding the ore onto the cathode plate, while outside of the other side wall of the box *P*, towards which the cathode rotates, there is fixed a vertical scraper plate *S*, adjustable by set screws *s s*. Finely ground galena of about 70 to 80 per cent of lead is fed continuously at about the rate of ten pounds per hour for a current of 1500 amperes, distributed so as to provide about 60 amperes per square foot of active cathode surface. The width of the feed slot is so adjusted that the total amount of ore resting on the cathode at any one time is about 15 pounds, the layer of ore being 1-10 to 1-30 inch thick. The rate of rotation is about one turn in an hour and a half. The electrolyte is a 10 per cent sulphuric acid solution.

Electrolytic Apparatus.—C. P. Townsend, Washington, D. C. Patent 779,383, January 3, 1905. Application filed April 12, 1902; divided and refiled May 10, 1904.

Electrolytic Process.—C. P. Townsend. Patent 779,384, January 3, 1905. Application filed April 12, 1902; divided and refiled May 11, 1904.

The process is intended especially for the electrolysis of aqueous solutions of alkali metal salts, and its distinguishing feature is the liberation and recovery of the products formed at the cathode, in the presence of a liquid which is substantially immiscible with and inert towards the products. The apparatus, as applied to the electrolysis of an aqueous solution of sodium chloride with production of caustic soda and chlorine, is shown in transverse vertical section in Fig. 3. It comprises a central anode compartment 1 and two cathode compartments 3, 3. The anode compartment is filled with the sodium chloride solution, a continuous flow from below upwards being maintained by means of pipe 11 and outlet 12. The anodes 2 are plates of Acheson graphite, carried by a conductor bar 2' of similar material. Diaphragms 5 of asbestos or other suitable pervious material separate the anode compartment from the cathode compartments. They are supported between the perforated sheet metal or wire gauze cathodes 4 and the plates 6, which latter have numerous small perforations, except at their upper portions. These plates may be of hard rubber, earthenware, hardened asbestos or other suitable non-conducting material. Plates 7, 7, preferably of iron, are placed parallel to the cathodes in such a manner as to provide a comparatively narrow space between them and the cathodes, while leaving a relatively wide space towards the outer wall of the cathode compartments. Coils 8, 8 are introduced for heating or cooling the liquid in order to maintain its temperature at any desired point. The liquid in the lateral compartments should be of such character as to separate readily from the caustic soda and to be so inert with respect to the caustic that it will not undergo any rapid, injurious change or modification in use. As such liquids, the non-saponifying oils are mentioned. The sodium set free at the cathodes is in part or entirely oxidized by the solution in the diaphragm, or percolating through the diaphragm, the caustic soda solution thus formed under the oil detaching itself freely from the cathodes. The hydrogen resulting from the reaction of the sodium upon the water, in passing upwards through the narrow spaces between the cathodes and the plates 7, induces a strong circulation of the oil, which further aids in detaching the globules of caustic soda, and thus removes them quickly from the field of electrolytic action and prevents their diffusion into the anode compartment. It is stated that current densities equal or exceeding one ampere per square inch of cathode surface are utilized with a high degree of efficiency.

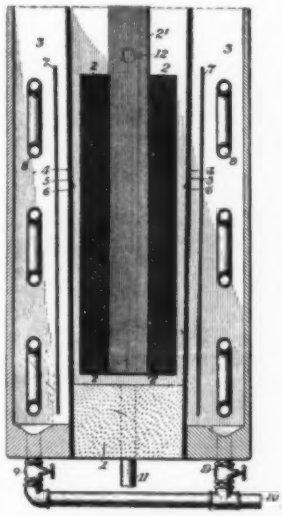


FIG. 3.—DIAPHRAGM CELL FOR ELECTROLYSIS OF SODIUM CHLORIDE.

Process of Electrolytically Refining Lead.—A. G. Betts, Lansingsburg, N. Y. Reissued patent 12,301, January 3, 1905. Original 713,277, November 11, 1902. Application for reissue filed April 22, 1903.

Patent 713,277 was described in ELECTROCHEMICAL INDUSTRY, Vol. I., page 133, while the process was mentioned repeatedly, and also fully described by Mr. Betts in ELECTROCHEMICAL INDUSTRY, Vol. I., page 407. In the present specification, he states that in practicing the invention he prefers a current strength of ten to twenty amperes per square foot of cathode surface, and a corresponding tension of .15 to .35 volts for each element (presumably cell). With higher currents the lead becomes harder and more brittle, and takes on a whitish color and a silvery luster. In general the deposited lead is apt to be slightly stiffer than ordinary lead, but the stiffness may be regulated at will by varying the current density and the quantity of organic reducing agent, such as gelatine, etc., in the solution, one part of gelatine, by weight, to five thousand parts of solution having been found preferable. It is also claimed that the process is well adapted for electroplating metals with lead, as the deposit obtained is a firmly-adherent, dense coating.

STORAGE BATTERIES.

Process of Making Spongy Lead for Storage Battery Elements.—C. J. Reed, Philadelphia. Patent 778,894, Jan. 3, 1905. Application filed May 7, 1903.

In carrying out the process one or more compounds of lead

in a powdered condition, such as the oxide, chloride or sulphate of lead, are mixed with finely divided metallic zinc or zinc dust, in chemically equivalent proportions. This mixture is subjected to the action of liquid of such composition as to decompose the lead salt into metallic lead, the acid radical uniting with the zinc to form a soluble zinc salt, which is then leached out with water. Among the materials which may be used are red lead and zinc dust in chemically equivalent proportions, and a solution of zinc chloride. The mixture of powdered materials may be treated with the solution while in a loose form to produce the finely divided lead for subsequent application to a battery plate. The mixture may also be slightly dampened and applied to the plate or battery frame as a solid coherent mass. It is then dried out sufficiently to make it suitably absorbent and subjected to the reducing liquid until the composition has been completely converted into metallic lead and a soluble zinc salt. It is then finally subjected to a water bath for a sufficient length of time to remove the zinc salt, and the plate is then ready for use in a storage cell. It is sometimes found advisable to mix with the powdered material a soluble, inert, granular material, such as sugar, salt or zinc sulphate, which may be readily dissolved out after the conversion of the lead compound to metallic lead has taken place, thus leaving pores for the entrance of the battery electrolyte.

Protective Covering for Storage Battery Plates.—Achille Meygret, Paris, France. Patent 779,553, January 10, 1905. Application filed April 6, 1904.

The coating described in the present specification embodies an improvement as compared with those described in the inventor's preceding patents (Electrochemical Industry, Vol. III., page 31; Vol. II., page 206), inasmuch as it is not applied directly to the active material of the plate, which might thereby be subjected to ill effects, but a film of a different material is interposed between the active material and the coating. This material is of a soluble nature, so that, when the battery is in use, it will be dissolved by the battery liquid, thus leaving the protective coating still surrounding the active material of the plate, but not firmly adhering to it. The substance preferably used for this intermediate film is a syrup formed of common lump sugar or saccharose, or glucose dissolved in water to such a consistency as to furnish a bath, into which the grid or metal of the plate, with the active material upon it, can be dipped. The syrup adheres to the surface and forms a film, the plate being then dipped into the solution, which is to give it the protective coating, and which solution consists of tetraacetate of cellulose or tetrabutyrate of cellulose or castor oil, essence of turpentine and nitrate of cellulose. The protective coating is not affected by the battery liquid, but the syrup film is dissolved out as described above.

Galvanic Elements.

Electric Battery.—W. H. Gregory, Vallejo, Cal. Patent 777,851, December 20, 1904. Application filed May 14, 1904.

The invention relates to a battery of the dry type; provision is made in its construction for resupplying the cell with moisture, after it becomes exhausted. For this purpose a glass tube is embedded within the cell, which lies against the inner surface of the zinc or metal casing, and is filled with water or any desired liquid. When the battery is exhausted, this tube is broken by striking the casing.

Electric Battery.—A. Gabrielson, Milwaukee. Patent 778,653, December 27, 1904. Application filed July 21, 1904.

The battery is a dry cell for open circuit only, and consists of a zinc cup coated with mercury on its inner surface. The cup is provided with a lining of paper soaked in dilute sulphuric acid. The cell is filled with a mixture of coke dust, powdered graphite and sal-ammoniac crystals, moistened with potash solution. This mixture is packed around a battery carbon, serving as the other electrode.

Galvanic Battery.—C. J. Reed, Philadelphia. Patent 778,893, January 5, 1905. Application filed July 12, 1901.

The battery is divided into two compartments by a porous partition, preferably of zigzag form. One compartment contains a solution of one or more ferrous salts, such as ferrous chloride, or ferrous sulphate, or a combination of two or more of such salts, while the other compartment is filled with a solution of one or more ferric salts, such as ferric chloride or ferric sulphate. In the ferrous electrolyte there is located an iron electrode, while the electrode in the ferric solution is electronegative to iron, and may consist of carbon, platinum, platinized carbon, etc. When the battery circuit is closed, the iron electrode becomes the anode and dissolves, thus forming a ferrous salt, while the ferric salt in the other compartment is also changed into a ferrous salt. This solution may be regenerated by oxidizing agents, such as oxygen, chlorine, bromine, ozone or hydrogen peroxide, preferably blown through the solution. When the battery is exhausted through exhaustion of the ferric salt, it may be regenerated by passing a current in the opposite direction through the electrolytes, which action deposits iron from the ferrous salt upon the iron electrode and reconverts the ferrous salt in the other compartment into ferric salt. When regenerated in this manner, the battery becomes an accumulator. In that case an electrode is preferably employed in the ferrous salt, which has a base of copper, carbon or other suitable electronegative substance, upon which a coating of iron has been electrolytically deposited. The battery may then be discharged until nearly all the iron is dissolved, and may be recharged without any substantial alteration in the form of the electrode, whereas with an iron electrode the latter might be dissolved irregularly, and would be unlikely to retain its original form when subjected to a series of charges and discharges.

Dry Cell.—Walter Stoeckigt, Greiz, Germany. Patent 778,912, January 3, 1905. Application filed May 12, 1904.

The cell is provided with a filling and replenishing tube for the purpose of introducing the electrolyte, and replenish it whenever desired. The tube has a number of perforations distributed along its length.

Electric Battery.—F. P. Dewey, Washington, D. C. Patent 779,589, January 10, 1905. Application filed March 25, 1902, renewed May 19, 1904.

The battery comprises a zinc plate supported from the top of a battery jar by a thimble of gutta percha or similar material. Within this thimble and above the zinc plate is placed a porous cup, upon the bottom of which rests a piece of asbestos cloth, which in turn carries the other electrode, consisting of a coil of copper or nickel ribbon. The latter is wound in the form of a spiral. The cathode is not immersed in the electrolyte, the latter being brought to its lower edge by capillary action of the pores of the diaphragm and allowed to spread in a thin film over its whole surface. Polarization at the cathode is claimed to be overcome by this means.

Battery Zinc.—F. J. Delarie, Pittsburg. Patent 779,693, January 10, 1905. Application filed July 1, 1904.

The invention relates to means of supporting the battery zinc, which consist in a bridge piece of stamped sheet metal resting on top of the battery jar, provided with shoulders and with a central key slot adapted to receive and securely hold the stem of the zinc electrode.

Battery.—N. M. Watson, Detroit. Patent 777,985, December 20, 1904. Application filed June 13, 1904.

The invention relates to a battery more particularly designed for electrotherapeutic purposes. The construction is therefore simple and compact, in the shape of a cylindrical casing containing a dry cell and the induction coil. No external wires are necessary, and the ends of the casing need only be grasped by the hands if it is desired to subject the system to the effect of the induced current. The construction of the cell is described in detail.

Thermo-Electric Element.—A. L. Marsh, Lake Bluffs, Ill. Patent 779,090, January 3, 1905. Application filed June 4, 1904.

The inventor states that he has found that the chromium

group of metals, comprising chromium, molybdenum, tungsten and uranium, when alloyed with nickel, may be formed into elements, strongly electronegative to elements composed of an alloy of nickel and copper. The positive element is preferably made of an alloy of 35 per cent nickel and 65 per cent copper, which has a fusing point much above 1050° C., the approximate fusing point of copper. An alloy of chromium and nickel, or molybdenum and nickel, the nickel predominating in each case, is preferably employed as the negative element. Any of the alloys of nickel and a metal of the chromium group is electronegative to the alloy of nickel and copper, and a bar of either may be readily welded to a bar of the other to form a highly-refractory non-oxidizable, tough, durable and efficient couple, which may be subjected to a concentrated, intense heat, without danger of injury.

MISCELLANEOUS.

Production of Paper Pulp. I. Kitsee, Philadelphia. Patent 775,829, Nov. 22, 1904. Application filed May 1, 1903.

The object of the invention is to effect an improvement in the present process of extracting the resinous and other intercellular matters from wood fiber which is to be used for the manufacture of paper pulp, which extraction, up to the present time, has been accomplished by boiling the wood in caustic alkali with addition of caustic lime and carbonate of soda. For this purpose it is proposed to use the ordinary digester as used for the treatment of wood chips, with such alterations as are necessary to carry out the process outlined below. The boiler shell is connected into an electric circuit as the cathode, and a structure is arranged in the center composed of a series of pieces of porous pipe, superposed upon each other and cemented at the joints so as to constitute a central tubular compartment. An anode consisting of platinum or carbon is introduced into this compartment and the whole of the apparatus is filled with a strong solution of common salt, after the outer compartment has been filled with the wood chips or fibers intended to be treated. Steam or another source of heat is then applied and the current passed through the apparatus, with the result that alkali will be formed in the outer compartment, which will extract the resinous and intercellular matter from the wood, while chlorine is given off in central compartment. It is proposed to use the chlorine for bleaching the pulp coming from the digester, after it has passed through the usual process of washing. It is stated that a current density of about 100 amperes for a digester of say 8 feet in diameter by 6 feet in depth is sufficient for all practical purposes, but where a stronger alkalinity is required in a shorter time the density should me raised accordingly.

Apparatus for the Treatment of Gases.—E. A. Werner, St. Louis. Patent 777,987, December 20, 1904. Application filed June 13, 1904.

Process of Generating Gases from Air.—Same inventor. Patent 777,988, December 20, 1904. Application filed August 6, 1904.

The apparatus consists of a chamber with an insulating bottom, on which are supported two current terminals, formed of thin strips of metal, between which an arc is struck. A gas supply pipe, by means of which compressed air can be introduced into the chamber, terminates below the arc, which is by this means deflected and elongated in a lateral direction, such deflection varying with the pressure and amount of the air supplied. At the same time, the latter serves the purpose of cooling the terminals. A further regulation and control of the back pressure in the apparatus and the amount of deflection of the arc is furnished by outlet valves on the conduits which conduct the treated gas to the various points where it is to be used. The second patent protects the process of carrying out the above operations.

Apparatus for the Treatment of Gases.—Same inventor. Patent 777,989, December 20, 1904. Original application filed June 13, 1904. Divided and refiled August 11, 1904.

Apparatus for the Treatment of Gases.—Same inventor. Patent 777,990, December 20, 1904. Original application filed June 13, 1904. Divided and refiled August 11, 1904.

Device for projecting air against electric discharges.—Same inventor. Patent 777,991, December 20, 1904. Application filed August 29, 1904.

The apparatus described in the first patent is the same in principle as that described above, except that a conduit leads from the electrode chamber to a cylinder in which is located a piston held in position by a spring. The piston rod is connected with a switch lever, which in turn cuts in or out, as the case may be, a certain number of sections into which the secondary coil of the transformer is divided. Accordingly, as the pressure in the apparatus rises or falls, a decreasing or increasing number of sections is cut into the circuit, and the amount of current supplied to the arc is thus correspondingly automatically lessened or increased. The second patent provided means for cooling the transformer by locating it in a chamber situated below the treatment chamber and passing the air to be treated around it before it passes through the arc. The third patent protects the device for injecting the air, so as to divert the course of the arc and to cause it to spread apart, for which purpose the supply pipe is covered by a cap provided with a number of small holes. The pressure of the compressed air is preferably from twenty to fifty pounds.

RECENT METALLURGICAL PATENTS.

IRON AND STEEL.

In our Vol. II., page 503, an account was given of James Gayley's now famous large-scale experiment of the application of dry-air blast to iron production. The air is dried by artificial cooling, a patent to this effect being mentioned in our Vol. II., page 469. A new patent of Mr. Gayley (779,037, January 3) gives some further details which throw light on some causes of the great success of his method. When the air is first dried, and the moisture reduced to a small and uniform percentage, it is, of course, possible to reduce the percentage of the coke employed in the charge by an amount equal to what would have been required to dissipate the moisture from the air. For instance, when the moisture contained in the air has been lessened from an atmospheric content of 6 grains of water per cubic foot of air to 1.5 grains per cubic foot, the saving of fuel calculated from the heat required to dissipate the 4.5 grains of eliminated moisture is about 7 per cent. Mr. Gayley has found, however, that if he goes further and reduces the fuel by 16 to 20 per cent, he not only maintains the furnace in successful running order, thus saving the cost of the fuel, but he improves actually the quality of the iron by lessening the extent to which it is contaminated with sulphur and phosphorus; he also obtains greater uniformity of work, prevents waste of ore carried away from the furnace by the gases, reduces the temperature of the gases leaving the throat of the furnace, and increases the output of metal about 20 per cent. The inventor attributes "these results, in part, at least, to the increased reducing power imparted to the furnace gases by the removal of the moisture from the air-blast; but it may be due in part to other causes." Some notes on critical remarks by foreign authors with respect to this subject are given on another page of the present issue in the Synopsis of Periodical Literature.

J. A. Mathews (779,171, January 3) patents "high-speed

steel containing not less than 0.10 per cent of vanadium." (First claim).—High-speed steel is defined as steel hardened by air and containing, in addition to carbon, either molybdenum or tungsten, either alone or in combination with chromium. Vanadium and ferro-vanadium has proven extremely attractive to steel men in recent years, though very little exact information is so far available as to the action of vanadium in steel. The present inventor adds about 0.50 per cent of vanadium to high-speed steels, containing from 6 to 10 per cent of molybdenum (or in the alternative from 12 to 20 per cent of tungsten) and from 0.5 to 1 per cent of carbon. He states that by the addition of vanadium, "the efficiency and endurance of the tool were more than doubled; there was also a great increase in the resisting power of the tool as against the sudden cross-sectional strains and shocks which are not infrequent in severe usage.

C. W. Cowen (779,074, January 3) patents details of construction of a frame for covers for crucible steel-melting furnaces.

F. A. Reynolds (780,128, January 17), in a brazing compound for welding cast iron, makes use of the reducing properties of aluminium. Equal parts of aluminium powder and carbon are mixed with oil or water, so as to form a paste. This paste can be used for painting the ends of the parts to be united, after which these ends are placed in the fire and heated to a red heat. After this borax and the ordinary brazing compound or spelter can be melted on over the joint, and allowed to cool. These parts are then filed off, as usual. "The aluminium has so great an affinity for oxygen that it reduces all other oxides present, and leaves an absolutely metallic surface at the parts which are to be united, so that there is nothing to prevent a perfect union." He prefers to use the brazing compound in stick form; the ordinary spelter or brazing mixture, a quantity of aluminium powder, and some oxide of iron are mixed together with a little copper to toughen the material. This mixture is then heated up to a red heat, when it melts down and forms an alloy. This alloy, in the form of sticks, is applied directly to the heated ends of the parts to be brazed.

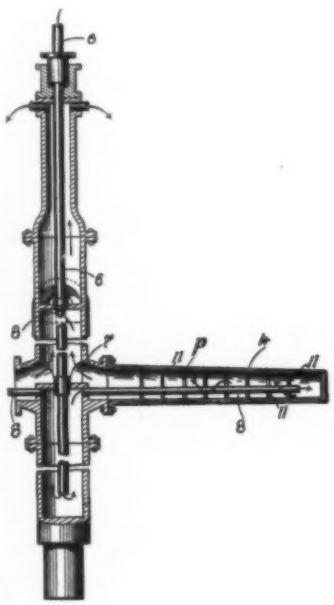


FIG. 1.—CONSTRUCTION OF RABBLE-SHAFT AND ARMS.

Iron and Zinc.

S. Peacock (779,310, January 3) proposes to treat ores containing sulphides of iron and zinc as follows. The ore is pulverized and roasted. The roasted ore is then either heated in a furnace while mixed with carbon or subjected to the action of a reducing gas in a furnace at such a temperature that while the iron oxides are reduced to metallic iron the zinc oxides remain unreduced. After all the iron has been practically reduced to a metallic condition, the furnace is cooled down in a reducing atmosphere. The metallic iron is then removed from the ore, either by a magnetic separator or by gravity. The oxide of zinc which is then practically pure, so far as the presence of iron is concerned, is then subjected to the distillation process in the usual way.

Roasting Furnaces.

Frank Klepetko (779,717, January 10) patents air-cooling devices for the McDougall type of ore-roasting furnace. Fig. 1 shows a section of the rabble-shaft and arms. The air-feed pipe 6 is located within the shaft and extends to a short distance from the closed bottom of the latter, the lower end being open, so that air blown through six discharges into the shaft, near the bottom, as indicated by the arrows. The shaft itself is divided into a series of chambers, separated from one another by the transverse partitions 7, slightly above the bottom of the adjacent rabble-arms 4. The air is thus forced from the shaft into the pipes 8, which extend into the hollow arms 4. The inner opposite wills of these arms are provided with an alternating series of deflecting ribs 11, the free edges of the ribs being concave, whereby the concavities of the entire series of ribs from an opening for the free passage of the pipe 8. The air, when discharging from the end of 8 into 4, is then forced to pass through the space between the walls of 8 and 4 in a zig-zag way, and is then discharged into the next higher chamber of the hollow shaft. In this way the cooling air is forced to come sufficiently long into contact with the surfaces to be cooled.

A. R. Meyer (780,115, January 17) also patents cooling devices for the McDougall type of roasting furnaces. For cooling he employs air charged with moisture, water flowing downward along the inner face of the rabble-shaft, while air is circulated in such a manner that the vapor rising from the heating of the water is carried by the air in contact with the shaft, and with other parts to be cooled. Water is introduced at the upper end of the shaft from a supply pipe, and is discharged into an annular gutter within the shaft near its tow. Similar gutters are provided at intervals, so as to hold limited amounts of water along the line of shaft. The connection between the shaft S and the hollow rabble arms is shown in Fig. 2. Into the hollow arms a current of air or air charged with vapor, is blown through 14 from the supply pipe P, centrally located within the shaft. The plate separates the interior of the shaft from the interior of the rabble arm, and the latter space is partly divided by the partition 22 into a lower and upper half. The cooling air introduced through 14 passes through the lower half to the end of the rabble-arm and returns through the upper half to be discharged through 18. In order to avoid the overheating of the necks or bosses, some of the water which flows downward is discharged around the inner part of the bosses.

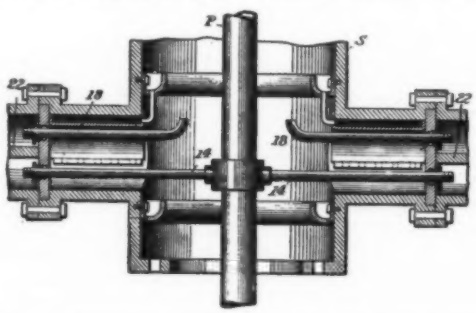


FIG. 2.—CONNECTION BETWEEN SHAFT AND RABBLE-ARM.

H. C. Davey (779,119, January 3) patents mechanical details of construction of a roasting furnace of that class in which a horizontal or slightly inclined cylinder is revolved upon its axis, the ore passing gradually from one end to the other. The inventor provides a series of revoluble inclined barrels, each having stationary heads and furnaces connecting therewith, through which heat is delivered to pass through the

barrels. These barrels are set with relation to each other and provided with intermediate automatic discharge chutes from the end of one cylinder, and receivers into which the ore passes and by which it is delivered to the succeeding cylinder.

RECOVERY OF METAL VALUES IN SLAGS AND MINE WATERS.

R. Baggaley (779,252, January 3) patents a process for recovering the copper held in solution in mine waters—as, for instance, those discharged from the mines in the Butte district, and for recovering simultaneously the copper, gold and silver values in the ferruginous copper-smelter slags commonly called in the trade "foul slags." From the latter he separates by a special smelting process, the iron which is drawn from the separating furnace and granulated by pouring it in a molten condition into water. This granulated iron is an excellent agent for precipitating the copper from the mine waters. These waters are passed through successive vessels containing the iron and are caused to drop from one vessel to the other; the copper is thereby precipitated.

CYANIDE PROCESS.

T. B. Joseph (780,293, January 17), patents a modification of the ordinary cyanide process for the extraction of metals from ores. His leaching solution is a mixture of "water, sodium cyanide, bicarbonate of ammonium, bromo cyanide, barium dioxide, and hydrate of calcium" while compressed air is forced upwardly in the tank from the bottom. "The cyanogen and the cyanogen bromide in this solution dissolve the precious metal, while the ammonium dissolves not only some silver, but also the copper, nickel and zinc, if there is any therein, and thereby causes a better extraction of the silver, as well as of the gold, with this solution than the straight cyanide process would obtain, as the ammonium therein holds the most of the dissolved copper, zinc and nickel leached from the ore, and prevents them from occupying so much of the cyanide as they otherwise would. When it is desired to extract much copper, silver, nickel, or zinc, as well as the precious metals from the ore with this solution, one should increase the quantity of the ammonium bicarbonate therein accordingly; but when it is only the precious metals in the ore to be extracted then only a small quantity of the ammonium bicarbonate need be used." As a general rule, the author says that there should be used in proportion to one ton of the water of solution, about 2 pounds of sodium cyanide, 1 pound of ammonium bicarbonate, enough of the bromo-cyanide solution to contain about 2 ounces of bromine, about 4 ounces of barium dioxide, and enough of the calcium hydrate to contain about 1 pound of the oxide of lime before being dissolved in the water.

L. E. Porter (778,547, December 27), treats cyanide solutions containing precious metals and copper as follows. He first precipitates gold and silver by means of zinc shavings; this leaves a solution of copper cyanide, which is subjected to the action of zinc dust and ammonia precipitating metallic copper, and leaving an alkaline cyanide solution suitable for further use. The process is said to be particularly applicable when the solution already contains lime. The resulting solution is used to leach a further quantity of ore, and in this operation the zinc cyanide and alkaline cyanide are all effective in dissolving the copper, silver and gold of the ore. By a continued repetition of this process zinc cyanide would, of course, accumulate in the regenerative solution; to overcome this, sufficient carbonate of soda is added from time to time to precipitate a portion of the zinc, as a double carbonate of zinc and lime, which, being insoluble, is precipitated. The inventor says that the process has been sucessfully used in practice, and that "the following theoretical conditions explain the operativeness of the process. When the alkaline or free cyanide is completely saturated with the metal cyanides which are known to be good solvents for metals—especially the cyanides of zinc and copper, which are good solvents for gold and silver—then ammonia will cause the metals to precipitate in the presence of zinc. The ammonia acts in the place of free cyanide. The zinc dust used is an impalpable powder, and each particle seems to be entirely consumed with the first precipitation of copper. The zinc-shavings are comparatively coarse, and after the surface has been entirely coated over, first by copper and later by gold and silver, there still remains a mass of zinc, which continues to precipitate gold and silver from the copper-cyanide solution without further precipitation of copper. The general operation may therefore be assumed to be as follows: The copper is first precipitated on the zinc-shavings, and is then dissolved, being replaced by gold and silver, and the surface exposed by the zinc-shavings should be sufficient to enable precipitation of all the gold and silver from solution by the time that the zinc-shavings are full coated. When all the gold and silver has thus been deposited, the addition of zinc in the form of dust exposes a far larger amount of surface, so as to precipitate all of the copper present.

I. Anderson (778,348, December 27), patents a process of recovering precious metals from solution, based upon the fact that when sulphuric acid is added to cyanide solutions of the precious metals in which sulphides and chlorides are present, the sulphuric acid so added will combine with the potassium of the potassium cyanide to decompose the double cyanides formed by the action of the potassium cyanide on the precious metals. The precious metals freed from combination with the potassium cyanide will then combine with the sulphur and chlorine to form insoluble sulphides and chlorides, which will be precipitated, leaving the cyanogen in the liquid in the form of hydrocyanic acid or in combination with the radicals with which the sulphur and chloride has been combined. The precious metals having been precipitated as chlorides or sulphides, the supernatant solution may be decanted off and the potassium cyanide regenerated by decomposing the potassium sulphate formed by the action of the sulphuric acid and removing the sulphuric acid from the solution. This decomposition of the potassium sulphate and the removal of the sulphuric acid is accomplished by introducing lime into the solution decanted off the precipitated gold and silver values, the lime so added causing the decomposition of the potassium sulphate with the formation of insoluble calcium sulphate, and the simultaneous regeneration of potassium cyanide.

BRIQUETTING.

U. Wedge (780,464, January 17), patents a method of briquetting iron pyrites fines for desulphuration. He has formerly patented a method of briquetting with sulphate of iron as a binder. He now states that iron pyrites reduced to a powder and mixed with sufficient water to render it plastic will, when subjected to air and especially under the influence of heat, rapidly oxidize and produce iron sulphate which acts as binder so that they may be easily pressed into the form of briquettes.

A. Ronay (778,899, January 3), patents a process for briquetting finely granulous ores, or the like by the following method. The powder or granular material is subjected in a dry or slightly damped condition to a pressure of at least 800 atmospheres up to 2000 atmospheres in such a manner that the pressure is gradually increased for the purpose of letting the air completely escape. The highest pressure at which material becomes plastic is applied only at the last stage of the pressing process, whereupon the briquettes are further subjected to the action of carbonic acid gas. The apparatus for applying the pressure is described.

MISCELLANEOUS.

A. E. Manchester (779,953, January 10), patents details of construction of a smelting furnace, the feature being the provision of a new cut-off valve which is interposed between the furnace and the fore-hearth, whereby the furnace and the fore-hearth nozzles may be brought into open communication when the valve is in one position, and be closed when the valve is in another position.

E. H. Carroll (779,412, January 10), patents details of construction of a billet-heating furnace of the type comprising an air-heating oven by which the air fed to the heating chamber as a constituent part of the gaseous fuel is heated. The feature of the construction is the provision of a supplementary flue between the heating chamber and the stack, independently of the air heating oven, and the provision of suitable dampers for controlling the products of combustion.

T. B. Parkison (779,307, January 3), patents a process of manufacturing mineral wool. Ordinarily, mineral wool is produced by directing a strong blast of air or steam through molten slag as it issues from the cupola to blow the molten mass through a blow-tube and into a blow-chamber or receiver. The material settles in the blow-chamber in the form of a mass of fine fiber interspersed with considerable quantities of minute globules or beads, the presence of which is due to the imperfect conversion of the slag into a fibrous form. The inventor has found that the bead-like particles are produced by the cooling of the slag before the latter is completely converted into fibrous form, the minute beads solidifying at the ends of the filaments. He also found that by subjecting the material during its flight to the action of hot vapor, preferably smoke or a combination of smoke and steam, the fibers are softened or annealed, materially lengthened, and given additional tensile strength, with the result that the body of the wool when taken from the settling chamber is of a soft coherent texture, and is almost wholly devoid of beady particles.

F. C. Weber (778,345, December 27), subjects the mixture of powdered aluminium and an oxide, preparatory to the aluminothermic reaction, to a drying, dehydrating and briquetting process.

A. Custodis (778,846, January 3), patents a process of making coke, which consists in mixing coal with top dust of blast furnaces containing iron, lime, silicic acid, and alumina, grinding the mixture, compressing it into cakes, and coking the cakes, whereby a slag is formed and the iron is reduced to form a sponge.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

INDUSTRIAL APPLICATIONS OF ELECTROCHEMISTRY.

Lead Refining.—In our Vol. I., page 407, we published a long illustrated description, by Anson G. Betts, on his process of electrolytic lead refining; some notes on recent developments were given in our Vol. II., page 509. The January issue of *Mines and Minerals* contains an illustrated description, by R. L. Whithead, of the plant at Trail, B. C., as remodeled in 1903 by the author. Up to that time 28 tanks had been used; then forty-four new tanks were installed, differently arranged and with different electrical connections. The electrolyte contained 10 per cent free hydrofluosilicic acid, with 5 per cent of lead in solution as lead fluosilicate. The potential between the electrodes was 0.25 volt. The anodes were cast 1 inch thick and placed 4⅛ inches from center to center, each tank containing 20 anodes and 21 cathodes. The cathodes, 1-16 inch thick, were obtained by a deposit of 30 hours on a steel plate, lead-coated and paraffined. For making starting sheets a current density of 4 amperes per square foot of cathode surface was used, while the current density on the working tanks was 15 amperes per square foot. The tanks are 7 feet 2 inches long, 2 feet 6 inches wide, 3 feet 6 inches deep, inside measure, and are made of 2-inch selected fir, painted inside and out with P. & B. acid-proof paint. Circulation is obtained through 1½-inch hard rubber pipes from one tank to another, as is customary in the cascade system. The arrangement of the original twenty-eight tanks was in

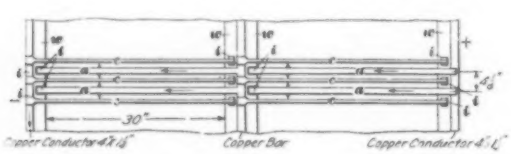


FIG. 1.—CONSTRUCTION OF ORIGINAL TANKS.

two rows of fourteen each, two tanks being placed side by side, connected by a small copper frame, the current entering the tank on a bar of copper, 4 inch x 1¼ inches, and leaving the second tank by the same size bar, as shown in Fig. 1, where *a* are the anodes, *c* the cathodes, *w* the tank wall and *i* wood insulators. The new forty-four tanks were arranged side by side, twenty-two in a row, each tank being connected with the tank ahead by thin sheet copper, 1-32 inch thick. The current, of course, enters on heavy copper and leaves the tank on same size copper as shown in Fig. 2, where *a* are the anodes, *c* the cathodes, *w* tank walls, *d* copper conductor, *e* copper between tanks 8 inches x 1-32 inch, *i* insulation. The circulation being through two tanks, instead of through fourteen, as in the arrangement of the twenty-eight tanks, the advantage gained here was the use of about 60 per cent less copper than in the old system, and gave a more uniform circulation. A rapid circulation of the electrolyte was found to be best, since it gave a more uniform percentage of lead in solution. It was found that as soon as the lead content at the bottom of the tank differed more than 1 per cent from the top, the deposit would begin to tree, thereby causing a short circuit.

The author then describes the melting house. The anodes weigh each 300 pounds and require six days before deposition is complete, the amount of scrap obtained being 22 per cent of the total weight of anode lead. By casting the anodes in closed molds and inserting copper hooks in the molten lead, so that they are supported by cross-bars instead of cast lugs, the amount of scrap can be cut down to 10 per cent. The cathodes weigh, when taken out of the tanks, from 150 to 200 pounds, and each plate is washed separately with hand brushes. During the sweating down of the cathodes in the refining lead kettle, about 4 per cent of dross is formed, which contains the antimony and tin that has been deposited; to insure the complete removal of both, the heat is raised until the lead shows red on the surface, then follows poling for 30

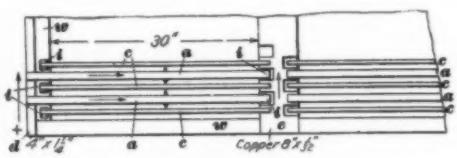


FIG. 2.—CONSTRUCTION OF NEW TANKS.

minutes, during which time the metal is constantly agitated and exposed to air, whereby any antimony or tin that has escaped the sweating process is removed, with the large amount of lead oxide formed.

Of special interest is the treatment of the slimes, and it seems that a perfectly satisfactory process has not yet been worked out. The slimes which adhere to the scrap of the anodes are removed from the tanks by lifts; the slimes are then washed off with stiff brushes. As the amount of lead

fluosilicate absorbed by the slimes during electrolysis is quite large, the percentage of lead and acid in the electrolyte is decreased from day to day, and necessitates that free acid and white lead be added each day to the electrolyte in order to maintain a uniform percentage of each ingredient. It was also necessary to add glue, about 2 pounds every other day, to each 2000 cubic feet of electrolyte; this prevents treeing and short-circuiting, and also gives a beautiful crystalline deposit, as shown in Fig. 3. After the slimes have been washed from the scrap, they are allowed to settle, and this first wash-water is run back into the electrolyte. To free the slimes from fluosilicic acid and lead fluosilicate, they are washed three times by decantation, with boiling water; after that, the slimes still contain 1 per cent of fluosilicic acid. The dilute solution of wash-water is concentrated by evaporation until it reaches a strength of 30° Be, giving by analysis 16 per cent metallic

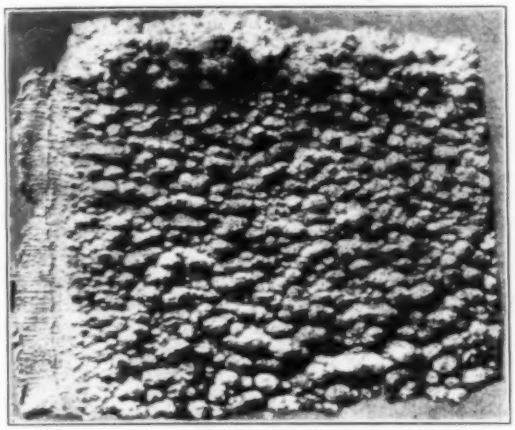


FIG. 3.—ELECTROLYTIC LEAD.

lead, and 8 per cent free acid; it is then run into a settling tank, cooled, and during the next twenty-four hours returned into the electrolyte in small amounts, so that it becomes thoroughly mixed. During electrolysis lead oxide is readily formed at the anode, and as readily goes in solution with the free fluosilic acid present, forming lead fluosilicate. A large amount of this salt is absorbed by the slimes, adhering to the anode, and does not enter the body of the solution, thereby depleting the lead content of the electrolyte. Another cause of further depletion of the lead content is the large amount of antimony in the slimes, from 25 to 30 per cent dry weight, only about half being soluble as oxide of antimony, Sb_2O_3, the remainder being combined with lead. This causes a further loss of lead in the slimes, and for each 10 tons of lead refined about 100 pounds are lost; this, in Canada, amounts to about 25 cents per ton. The author then discusses the characteristic advantages of the Betts process, which we have already repeatedly discussed. He finally discusses in great detail the methods now used for the refining of the slimes; this reference may suffice, since the author remarks himself that these methods are still incomplete, for they do not recover the impurities completely. He is now perfecting a process "in which electricity plays a prominent part," and which is intended to solve the question of melting silver in the presence of arsenic or antimony.

Calcium Cyanamide as Fertilizer.—The *Elektrotech. Zeit.* of December 29 contains some notes by Erlwein of the Siemens & Halske Co. on the production of calcium cyanamide, $CaCN_2$. The product is stated to be made a "rather (ziemlich) large" commercial scale in a plant in Germany, while a large plant for producing 4000 tons per year is being erected in Piano d'Orta in Italy. This product may be used as a cheap substitute for Chili saltpeter, and is stated to have proven satisfactory in extended fertilizer experiments made for three years by Wagner and Gerlach. There are two methods for making calcium cyanamide, the one starting from calcium carbide, invented by Moissan, Frank, Caro, Rothe, Pfleger (see our Vol. I., page 423), the other starting from carbon, lime and atmospheric nitrogen, invented by Erlwein (see our Vol. II., page 204). The German trade name of calcium cyanamide is Kalkstickstoff.

Ferro-Alloys.—In our Vol. II., page 309, we gave an illustrated description of the electric crucible furnace of P. Girod for making ferro-alloys and special steels. In an article by Schmidhammer in the *Oest. Zeitschr. f. Berg-u. Hüttenwesen*, of Nov. 12, some analyses of ferro-alloys made by Girod are given as follows: Ferro-chromium for fine steels: 67.27 per cent Cr, 31.82 Fe, 0.41 C, 0.17 Si, 0.11 Mn, 0.18 Mg, 0.007 S, 0.003 P. Ferro-chromium for armor plates: 67.10 Cr, 26.81 Fe, 4.2 C, 0.61 Si, 0.47 Mn, 0.23 Al, 0.17 Ca, 0.31 Mg, 0.01 S, 0.02 P. Ferro-tungsten: 85.47 W, 13.94 Fe, 0.35 C, 0.13 Si, 0.09 Mn, 0.005 S, 0.009 P. Ferro-molybdenum: 80.80 Mo, 16.79 Fe, 2.27 C, 0.11 Si, 0.02 S, 0.007 P. Ferro-vanadium: 49.50 V, 49.15 Fe, 1.07 C, 0.09 Si, 0.07 Mn, 0.10 Ca, 0.009 S, no P.

Chlorination versus Cyanidation.—The discussion started by W. E. Greenawalt's articles, abstracted in our last issue, is continued in the correspondence columns of the *Eng. and Min. Journal*, December 29. P. Argall writes: "The substitution of chlorine gas, or even chlorine water, in place of chlorine liberated in a nascent state, directly in the barrel, is certainly nothing new, nor can the electrolytic generation of chlorine be considered new, even when generated in the barrel. One of Cassel's numerous patented processes takes in that feature, while the Greenwood and other processes cover about all that is left, and, so far as I know, they are all commercial failures. The electrolytic decomposition of salt outside the barrel as a means of providing the chlorine for the charge, has been in use in the largest plant of the 'mill combine' for about a year; but it has not been, up to this time, introduced into the other mills of the combination, so I infer it is not good enough." He then defends the cyanide process against various objections raised against it, and reiterates his statement that the chlorination process "cannot begin to compete with modern cyanide practice. One of the simplest proofs is that the tailing discharged from any of the chlorination plants, after being subjected to the double process of chlorination and concentration, will pay handsomely to cyanide." William H. Davis refers in the same issue to a case in which an electrolytic chlorination process mill was dismantled and refitted for cyanide; "the electrolytic chlorination process, though preceded by roasting, had failed to extract the gold from ore which was a simple telluride." He then contradicts the view that neither the acid nor the bleach has any indispensable efficacy in the chlorination process, other than to deliver chlorine; he thinks that this view "overlooks the difference between molecular and nascent chlorine."

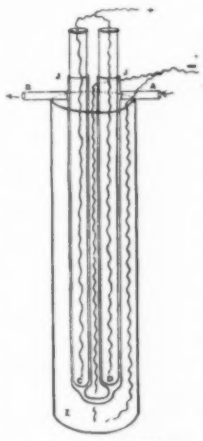


FIG. 4.—OZONIZER.

Ozone.—To the London *Elec. Review*, of December 30, J. B. C. Kershaw contributes another article on the production and utilization of ozone. He first describes a modified form of the Berthelot tube ozonizer, now being employed by Bone and Drugman in research work at the Victoria University in Manchester. This consists of a U-tube formed of two thin walled glass tubes, one inside the other, their surfaces nearly touching, and sealed together at *J*, as shown in Fig. 4. The outer U-tube has branches at *A* and *B*, for the inlet of air, and exit of the ozonized product. The conducting medium is a slightly

acid solution of copper sulphate, the inner tubes, *C* and *D*, and the outer thick glass containing vessel *E* being nearly filled with this solution. A stout copper wire passes down each of these tubes, and these wires are connected to the positive and negative poles of the induction coil or transformer supplying the current. The air passes along the very narrow annular space between the inner and outer tubes. The advantages of a liquid contact in place of metal are, that there is a more regular and silent discharge of the electric current in the space between the two glass tubes, and that the temperature of the apparatus can more easily be kept under control, a flow of the sulphate solution through the inner and outer tubes being very simple to arrange, if desired. Any number of single ozonizer tubes can also be grouped together, and for experimental work this form of ozonizer is said to be superior to any that has been yet tried. For industrial purposes, its fragile construction is, of course, a serious defect, and would probably lead to trouble. The author then describes the ozonizers of Elworthy and Vosmaer-Bebret, which were described in our Vol. II., pages 199, 153, 511, and then refers to the researches of Goldstein, who has recently discovered that it is the ultraviolet rays of light which-are-chiefly-instrumental in the formation of ozone. (This has been confirmed by others.—Ed.) Goldstein has designed a quartz-glass tube ozonizer which is said to utilize these ultraviolet rays to the fullest advantage. Two platinum electrodes are fixed in the ends of this tube, a vacuum is produced in it, and upon allowing the electric discharge to occur between the two electrodes, the tube becomes filled with the usual colored haze. Only the ultra-violet rays of light, however, can pass easily through the quartz-glass envelope which surrounds them. If the tube be now placed in an atmosphere of pure oxygen, cooled below 100°, almost the whole of the gas is transformed into ozone, and this condenses upon the outer walls of the quartz glass tube, as a blue liquid. Not only is the action of the silent discharge stated to be much intensified under these conditions, but the liquid ozone is said to be more stable than gaseous ozone, and less liable to be decomposed by the rays which produce it. The reaction, in fact, ceases to be a reversible one at — 100° C. The author then refers to Rosenberg's ozonizer noticed in our last issue, page 36, and gives the following table, which brings together all the hitherto published figures for the yields of the various forms of ozonizer, with the calculated cost of electrical energy, when the e. h. p.-hour is supplied at 3 cents:

COMPARATIVE YIELDS AND COSTS OF OZONE.

Form of Ozonizer.	Yield of Ozone in Grams per E.H.P. Hour.	Cost of Electrical Energy in Cents.	
		P'r Kilogramme of Ozone.	P'r Kilogramme of Active O.
Rosenberg	184	16.3	48.9
Yarnold	175	17.1	51.4
Otto	155	19.3	58.0
Andreoli	94	31.8	95.4
Elworthy	48	62.4	187.2
Siemens & Halske	25	120.0	360.0
Marmier & Abraham	20	150.0	450.0

The lowest figure in this list for the cost of the kilogram of active oxygen is still "greatly in excess of the cost by the usual oxidizing and disinfecting agent, namely, hypochlorite of lime (bleaching powder or bleach), namely, between 28 and 34 cents." It must also be remembered that the figures in the last two columns in the above table, give the cost of electrical energy only. The author then begins to make some remarks on the utilization of ozone for water sterilization, referring to the Marmier-Abraham system.

Theoretical and Experimental.

Electro-Analysis with Circulation of the Electrolyte.—The *Zeit. f. Elektrochemie*, of December 23, contains an account of experiments made by A. Fischer and R. J. Boddaert on electro-analysis with strong circulation of the electrolyte, produced, for instance, by rotation of the electrodes, in order to hasten the deposition. They experimented with nickel, zinc, copper, bismuth, cadmium, lead, silver, mercury, antimony, and tin. A perfect precipitate of nickel is obtained with rotating electrodes from an ammonium oxalate solution, the time of analysis being reduced from three hours to fifty minutes, with a hot solution, and to forty minutes with a cold solution. The method cannot be applied to Classen's oxalate method for zinc precipitation, since spongy zinc would be obtained; good results are, however, obtained with sodium acetate and acetic acid; the precipitate becomes best, and the result most exact with an alkaline solution with an addition of potassium cyanide, a nickel cathode being used. Copper is precipitated satisfactorily at a high rate from a nitrate solution with or without sulphuric acid; for precipitating the last quantities of copper, an addition of ammonia is necessary; the time is reduced from six hours to twenty minutes. With respect to bismuth, the authors found that strong stirring accelerates the precipitation from a nitrate solution and diminishes the formation of sponge without preventing it altogether; for this reason an exact analysis by this method is not yet possible. With cadmium, an oxalate solution is not suitable for precipitation at a high rate, since an increase of current density results in a spongy deposit; sulphate solution is more suitable, although the formation of sponge cannot be avoided altogether; in the latter case the time of analysis was ten minutes. By stirring with increased current density, the time of precipitating lead from a nitrate solution was reduced from one hour to fifteen minutes, the result being equally satisfactory. Strong circulation accelerates greatly the precipitation of silver from a potassium cyanide solution, but the quality of the silver is thereby impaired. Precipitating with stirring gives good results for mercury in a nitrate solution at ordinary temperature, and with not too high a current density; the time was thereby reduced from forty-five to fifteen minutes. In general, if the best methods are compared for both cases, the time required for precipitation of the metals investigated, with stirring and revolving cathodes, is four to eighteen times smaller than without stirring.

Electrolysis of Cobalt and Nickel Tartrates.—An account of experiments on this subject is given by J. E. Root in *Jour. Phys. Chem.*, January. In determining the apparent decomposition voltages under the conditions used in electrochemical analysis, the value of 1.75 volts was found for cobalt in an alkaline tartrate solution and 2.8 volts for nickel. However, the range of voltage in separating cobalt from nickel is not as great, since, in the presence of cobalt, the decomposition voltage for nickel in an alkaline tartrate solution is lowered. If the voltage be kept below 2.1 volts in separating cobalt from nickel, the metallic cobalt deposit contains no nickel; at higher voltages the cathode deposit always tests for nickel. While pure cobalt is obtained at voltages below 2.1 volts, it has proved impracticable to get all of the cobalt out of the solution in any reasonable time. One reason for this is the oxidation of cobaltous salt to a green cobaltic salt at the anode. Coehn's results as to the composition of cobaltic oxide, $Co_2O_3.3H_2O$, are confirmed by the author. The practical results may be summed up in the statement that the author found several methods for the electrolytic determination of cobalt alone, and that it is possible to obtain nickel and cobalt pure from an alkaline tartrate solution if one is willing to take the time and pains.

Calcium.—J. H. Goodwin, who has already published experiments on the electrolytic production of calcium (*Jour. Am. Chem. Soc.*, August, 1903) has recently presented another paper before the Amer. Philos. Soc'y., which is published in the *Proceedings*, Vol. XLIII., No. 178. The experiments were made at the laboratory of the University of Pennsylvania, and the method used was the electrolysis of fused calcium chloride. The furnace is shown in Fig. 5, where a bottom of

solid calcium chloride was maintained by the cooling effect of a copper coil *E*, through which is a thick piece of asbestos *C*, holding by the blocks *D*, the copper coil *E* well up in the Acheson graphite anode *F* and insulated from it by the asbestos *G*. The iron bands *H* conduct the current from the positive cable *I* to the graphite vessel *F*. The calcium grows and forms the stick *J*, which is started by the cathode *K*, connected with the negative cable *L* and supported by the clamp *M*, which is drilled and tapped at *N* to receive the screw *O* by which it can be raised or lowered. *P* is a tube sliding freely on the rod *Q* of the retort stand, and against which *R* is firmly screwed to make the clamp *M* fairly rigid without interfering with its vertical motion. Fluospar covers the copper coil and fills the space about it, while the furnace is filled with calcium chloride, which becomes solid at *S*, and is kept molten at *T* solely by the current passing through the furnace. The whole apparatus was set up inside an empty wind furnace, from which the grate bars had been removed. In this way the escaping chlorine was drawn from the room. Pure anhydrous calcium chloride was used, melted in a Dixon graphite crucible and added from time to time, but this was found to introduce much iron, aluminium and silicon from the clay binding used in the crucible, so that finally the chloride was added cold and melted by drawing an arc from the iron rod *K*. Thus, the furnace was filled sufficiently for a run. The anode was turned in a lathe from a 6-inch length of Acheson graphite electrode six inches in diameter. Being pure and able to withstand the high temperature and chlorine without disintegration, made this material by far the most suitable for constructing the furnace. In two experiments, the author succeeded in obtaining an ampere-hour efficiency of nearly 42 per cent, while the average ampere-hour efficiency was 29.1 per cent; the average volts were 17.7.

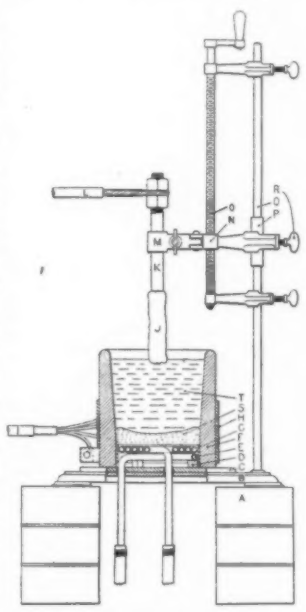


FIG. 5.—ELECTRIC FURNACE FOR PRODUCING CALCIUM.

The analysis of the calcium produced is Si 0.03 per cent, Fe 0.02, Al 0.03, Ca 98.00, Mg 0.11, Cl 0.90, O (by difference) 0.91. The author thinks that the difficulty experienced, until recently, in making metallic calcium, was probably due to the small scale on which the operation was tried. The simple and satisfactory operation of this furnace would lead one to believe that, technically, the process would be still more efficient and easily controlled. A furnace, five times as large, using about 1,200 amperes, would require about 8 volts, and the screw mechanism could be electrically controlled, keeping the current constant and the product perfectly uniform, as the rotary furnaces of the Union Carbide Co. are controlled. A water-cooled shield might be necessary to cool the large calcium cathode as it was drawn from the bath. The two essential conditions of operation are—first, rapid withdrawal of the metal formed to increase the yield and minimize recombination; second, narrow temperature limits. The bath must be hot enough to deposit the metal molten, not spongy, and cool enough to let it congeal upon the cathode and be raised without breaking off.

"The solid metal can be worked like other metals and is much more stable than imagined." It can be heated red-hot continuously in a triple Bunsen flame without igniting, but at this temperature its texture is like clay, and it can be easily squeezed apart with tongs, sometimes igniting at the edges and burning feebly till the lime formed smothers the flame. When cold, a bright calcium surface becomes dull rapidly in ordinary air, but if hot the metal can be brightened with a file or polished in the lathe with emery cloth and will remain bright as long as it is hot. The specific gravity is 1.5446 at 29.2° C., so that it is only four-sevenths as heavy as the light metal aluminium (specific gravity 2.68). Calcium is the fifth best conductor of electricity, being surpassed by silver, copper, gold and aluminium, if wires of equal diameter and length are compared; but for wires of equal weight and length the order is entirely different, calcium being second and exceeding silver by 67 per cent, copper by 62 per cent, gold by 86 per cent, and aluminium by almost 20 per cent. Calcium is harder than sodium, lead or tin, almost as hard as aluminium, but softer than zinc, cadmium or magnesium. The ultimate tensile strength of calcium is 8,710 pounds per square inch. A similar method of making calcium is understood to be used on a commercial scale at the Elektrochemische Werke of Bitterfeld in Germany (see, for instance, the paper of Rathenau, ELECTROCHEMICAL INDUSTRY, Vol. II., page 276). The metallic calcium is sold in form of sticks and is said to be consumed mostly in the iron and steel industries and for reduction purposes.

Action of Amalgams on Solutions.—The controversy between G. McP. Smith and G. Fernekes on the action of amalgams on various aqueous solutions and on the possibility of explaining the facts by the electrolytic dissociation theory is continued in the *Jour. Phys. Chem.*, G. McP. Smith contributing an article on the action of barium amalgam on solutions of sodium and potassium salts to the January issue. The author, who defends the dissociation theory, describes some new experimental facts and discusses at some length some points of the theory. "While all the phenomena of solutions may not be capable of explanation by the ionic theory, it is at present, nevertheless, very rash to claim it to have been shown 'conclusively' that this theory has outlived its usefulness."

Thermochemistry.—L. J. Henderson publishes a paper in the January issue of the *Jour. Phys. Chem.* on the heats of combustion of atoms and molecules. He shows that the heat of reaction of a substitution depends not alone upon the nature of the atoms or atom groups which take part directly in the reaction, but also in a high degree upon the chemical composition of the remainder of the molecule, and upon the arrangement of the atoms therein. Thus, the same substitution —H by OH—which leads from paraffine hydrocarbon to primary alcohol, with a loss of 40 calories to the heat of combustion of the molecule when it converts an aldehyde into an acid diminishes the heat of combustion by 72 calories. On the other hand, marked and parallel differences exist in the heat of formation from the hydrocarbons of primary, secondary, and tertiary alcohols, and of aldehydes and ketones. The heat of combustion of an atom chemically bound in a molecule is dependent not only upon its nature and the nature of the atoms with which it is directly united, but also upon the nature and position of every other atom of the molecule. If there are not present within the molecule considerable energy relationships of unknown character, the energy of a valence is variable, and a function of all the atoms of the molecule and of their positions.

METALLURGY.

IRON AND STEEL.

Fine Ores.—A standing order of the day, in the production of pig iron, is the discussion of obviating the difficulties caused by using fine ores, such as the Mesabi. It is probably the most serious question at present confronting American blast-furnace practice. Engineer Aloys Weiskopf, of Han-

over, discusses the question in a recent issue of *Stahl und Eisen*, November 1. He reviews the question of chemical equilibrium between FeO, Fe_2O_3, Fe_3O_4, Fe, C, CO and CO_2, and finds the most serious cause of the difficulties of smelting fine ores to lie in their very easy reducibility, by which they are reduced high up in the furnace, and the spongy iron thus produced has a large range of action during which it acts as a catalytic agent to cause carbon di-oxide to deposit carbon, while the finely deposited carbon clogs up the gas passages and causes "hanging" of the charge and explosions from accumulation of pressure and subsequent rapid oxidation of the fine carbon.

Desperate attempts have been made to overcome the difficulties by changing the lines of the furnace and the manner of charging, but the author thinks that the most hopeful solution lies in practicable briquetting processes. The sintering process is being operated at a cost of $0.75 per ton, in Germany. Very cheap fuel is gasified in gas producers, and the fine ore heated in furnaces to the point where it becomes sticky; this temperature is lowest for spathic ore, higher for purple ore, and highest for magnetite. The hot ore is then pressed at this temperature into briquettes, which are hard, strong and porous, while also free from water, sulphur, arsenic or carbonic acid, since the heating may be conducted as an oxdizing roasting. The process gives very good results, but requires careful supervision.

The slag-briquetting process consists in mixing 10 per cent of finely ground iron slag with the ore, making up briquettes and keeping them in a steam chamber for ten hours, at 120 pounds pressure per square inch. The action is a hydrochemical one, the slag being partly combined with the ore, under the influence of heat and moisture, and so causing a setting into a hard, porous briquette. The steam washes out some injurious ingredients, such as sulphur, but the slag added increases the work which the furnace must do in smelting the ore. This solution is not yet altogether satisfactory.

The Tunner Statue.—Of interest to all metallurgists, and particularly to iron and steel workers, is the unveiling of the beautiful statue to Peter von Tunner, at his beloved Bergschule in Leoben, Austria, on November 20. Numerious officials, dignitaries, colleagues and a host of students, miners and townspeople participated in the occasion, while the successor of Tunner in the chair of metallurgy at Leoben, Professor von Ehrenwerth, delivered a masterly oration. The statue consists of a granite pedestal and a bronze bust of Tunner, with life-size figures of a student on one side and a miner on the other, entwining a wreath about it.

Born in 1809, Tunner traveled on metallurgical journeys throughout most of Europe, in 1835 to 1838, and was named professor on the opening of the Mining Academy at Leoben in 1840. Here he taught, worked and lived until his death in 1897, at the age of eighty-eighty years. His best-known scientific works were his classical researches on the finery process, the puddling process, and the working of the blast-furnace. In thus honoring the memory of her celebrated metallurgist, his native land does honor to herself.

Verein deutscher Eisenhüttenleute.—The general meeting of this well-known German society of iron and steel metallurgists was held in Dusseldorf on December 4. The papers read were "On Large Gas Engines," by Prof. Eugene Meyer, of Berlin; "On Drying Blast for Furnaces by Refrigerating Machines," by Professor Carl von Linde, of Munich, and "On the Classification of Foundry Pig Iron," by Dr. F. Wüst, of Aachen. We will review these papers as soon as they reach us in full.

Dry Blast.—The unexpectedly valuable results attained by Mr. Gayley when using dried blast in a blast-furnace (see report of proceedings Iron and Steel Institute, Electrochemical Industry, December, 1904), have stirred up metallurgists in every direction to account for them by proper calculations. The point of this whole question is, that theory, so far as developed and understood, would account for a saving of only 2.2 per cent of the fuel by removing the moisture of the blast, whereas Mr. Gayley actually saved over 20 per cent. Professor Le Chatelier discusses this question in the *Revue de Metallurgie*, for December, and rather plainly intimates that some other conditions, not described in Mr. Gayley's report, must have caused the saving, or else that the results were deliberately falsified as a stock-jobbing scheme. The puerility of these objections would render them undeserving of notice, if they come from a less distinguished scientist; it is a pity that the critic could not have preserved a more judicial frame of mind. W. Schmidhammer, in *Stahl und Eisen*, for December 1, attacks the same subject, but in a much more judicial and judicious manner, and, in fact, does really come near to finding the correct solution of the enigma. He bases his calculation on the theoretical temperatures obtainable before the tuyeres, using dry blast and moist blast, and calculates them as 2465° C. and 2294° C., respectively. The calculations are not free from errors, but they show what is undoubtedly true, a much higher theoretical temperature before the tuyeres when using dry blast. This higher temperature, Schmidhammer argues, causes the furnace to work faster, extends the smelting zone, and so improves the efficiency of the furnaces as to account for the saving in fuel obtained. Schmidhammer is miles nearer to the correct solution of this question than is Prof. Le Chatelier, with his unwarranted insinuations, as is evident from the logic of Schmidhammer's remarks, and as will be abundantely proven by developments in the near future. By the latter we allude to a forthcoming discussion of this whole matter at the next meeting of the American Institute of Mining Engineers, at which it will be shown just what causes the improved working with dried blast, and where it will be shown that Schmidhammer has reached conclusions which are about half right.

Quenching Tests.—Le Chatelier has made a series of very interesting tests showing the rate of cooling of pieces of steel in various liquids and under varying conditions, with the resultant effects upon the hardness of the specimens. (*Revue de Metallurgie*, September.) The temperatures were taken by a thermo-couple embedded in the center of the specimens, and recorded autographically by means of a reflecting galvanometer and a movable photographic plate. The variations in rate of cooling were considerable; in 8 seconds after immersion, the temperatures fell from the 800° to 900° C. at starting to 40° C., when sprayed with cold water, 100° when dipped into water at 20°, 115° when dipped into cold brine, 160° in water at 35°, 190° in water at 50°, 190° in 10 per cent sulphuric acid, 460° in mercury, 550° in linseed oil, 670° in boiling water, and 725° in melted lead. The speed of cooling varies greatly with the temperature of the cooling fluid. Cooling in still air or in an air blast showed great variations; while a specimen cooled in cold water fell to 100° in 5 seconds, in boiling water to 320° in 5 seconds, and in oil to 320° in 17 seconds, or to 100° in 45 seconds, the pieces cooled in air remained at 700° at the end of 60 seconds, rising to 750° after that, by the effect of heat evolution at the allotropic critical point.

One practical result of these tests is to show the similarity of the cooling curves in oil and in hot water, showing the possibility of obtaining similar effects, which has already been proved in practice. The results obtained show the relative rates of cooling for bars of the size taken, viz., 18 millimeters square, and in these there is a certain relation between the rate at which the piece can conduct its internal store of heat to the surface and the heat-absorbing capacity of the surrounding fluid, the latter factor consisting of a combination of the specific heat of the surrounding fluid and its rate of heat conduction. For smaller objects, with the cooling surface relatively much smaller, the conductivity of the surrounding fluid plays a relatively much greater *rôle*, and the relative values of the fluids as hardening media are different

from those found by Le Chatelier. For small, fine steel objects, mercury is known to be a much greater hardener than water, which is the converse of Le Chatelier's results, and this is undoubtedly due to the greater influence exerted in this case by the larger heat-conducting power of mercury.

COPPER, BRASS AND BRONZES.

A Swede, Axel Westman, made a "metallurgical journey" into Germany, and his report to the *Jornkontorets Annaler*, translated into German in *Metallurgie*, of December 8, 1904, contains, among other things, the following information:

The Kupferwerke Deutschland, at *Oberschönweida near Berlin*, was erected in 1897, and is provided throughout with modern machinery. The copper used for copper ware is exclusively American electrolytically refined; for brass, Mansfeld copper and Silesian zinc is used. In the casting house, the crucible furnaces are sunk level with the ground, and the crucibles used carry up to 100 kilograms (220 pounds) of metal; coke is used as fuel, and blast is introduced by means of small nozzles just above the grates. The copper is first melted, then the scrap brass added, and finally the zinc, slightly warmed with vigorous stirring. The loss of copper is 0.5 per cent, and of zinc 6 per cent. The brass is cast into round ingots 3 inches in diameter by 36 inches long, in thick-walled molds, so as to chill the metal and make it stronger; it has been found that when using thin-walled molds, in which the metal set slowly, copper tends to sink to the bottom and zinc to accumulate on top. One article made on a very large scale at this works was rods of drawn brass, drawn with a section like the profile of a gear wheel. These rods are bored in the center and then cut transversely into thin sections, forming small toothed wheels for clocks and watches. Copper rods are rolled into forms with complicated cross-sections, which, when cut into sections in a similar manner form the copper blocks needed for commutator bearings. In casting brass into chill molds, a sort of sieve or filter filled with charcoal is put on top of the mold, which strains out dirt, oxide and other impurities, and divides up the falling metal into small streams. In rolling out sheet brass, the rolls become worn away more in the middle than on the edges. They are brought back to proper condition by screwing the rolls hard down together, and running them empty with a plentiful supply of water; they thus grind each other straight.

The Allgemeine Elektricitäts Gesellschaft has a large, newly erected copper and brass works at *Oberspree, Oberschönweide, near Berlin*. The casting house contains, in a row, thirty sunken crucible furnaces, and with fire-tipping crucible furnaces, in which the crucibles, holding up to 650 pounds, remain permanently until they are used up, after about 100 heats. When pouring ingots into chill moulds, it is usual to wash the mold with a wash of finely ground wood ashes. In this works, aluminium bronze containing 10 per cent aluminium was being manufactured, and cast into cylinders of 2 inches diameter, by 36 inches long. The copper used is the purest electrolytically refined, which is first melted, and then aluminium stirred in, resulting in a lively boiling, and the raising of the contents to nearly a white heat, a phenomenon ascribed by the author to formation of aluminide of copper [but which is in reality mostly due to the oxidation of some aluminium by dissolved cuprous oxide.—*Abstractor*]. The bronze thus made is so brittle that it cannot be worked either hot or cold, and it is only malleable after being remelted two or three times. It is general experience that the aluminium bronze is better the oftener it is remelted. The final casting is made in cast-iron molds, smeared with a mixture of graphite, pipe-clay and lard; and greatest care is taken that no skin of oxide runs into the mold. The waste is about 5 per cent. Finally, the aluminium bronze rod is placed in another mold and well deoxidized copper cast around it; the compound bar is rolled and drawn down to wire, which is sold as "Doppelbronzedraht" or "double bronze wire," used extensively in Europe for telegraph and telephone service, the bronze core giving great strength and the copper envelope high electric conductivity.

Phosphor bronze is made in this establishment by melting together a mixture of 93 per cent electrolytically-refined copper, 2 to 3 per cent of 10 per cent phosphor copper, and 4 to 5 per cent of tin; practically no phosphorus remains in the product. The melt is stirred with a copper rod well rubbed with powdered graphite, and cast into slabs for rolling. These slabs are rolled down to nearly circular discs of 3 millimeters thickness, by passing through the rolls in different directions; the nearly circular sheets are put into a circular cutter, which, starting from the outside cuts off a continuous strip 3 millimeters wide, forming, when straightened, a continuous rod 3 millimeters square, leaving a small circular waste in the middle. These strips are then rolled down to wire, which, after annealing, can be drawn down to the finer sizes. This phosphorbronze wire has an elastic limit which practically coincides with its ultimate tensile strength, and does not crystallize or weaken under rapid agitations of load. With these properties it makes a splendid material for wire rope used in mines, and particularly because it resists corrosion better than iron or steel.

The same firm rolls and draws down American copper wire bars to pure copper wire of 98 per cent guaranteed conductivity, using 21 heavy American drawing benches, with roller draw plates, which are described as the most perfect wire-drawing machines made. The finer wire, from 1.4 down to 0.05 millimeter diameter, is drawn through diamond dies, the larger sizes necessitating a 5-karat diamond for one die; the life of these dies is 6 months to a year. The hard-drawn trolley wire, of 8 millimeters diameter, made here has a tensile strength of 60,000 pounds per square inch and conductivity of 98.45 per cent.

The *"Letmather Messingwerk" at Letmathe* makes brass sheet and wire. There, brass is cast into slabs in sand molds, using 94 to 96 parts sand, mixed with 4 to 6 parts of washed clay, and made moist. The molds are dried in hot chambers above the crucible furnaces, which are kept hot by the waste heat of the latter. The brass contains 63 parts copper to 37 parts zinc, and is cast at as low a temperature as possible. The slabs are rolled to 5 millimeters thickness, the corners cut off, and then a spiral cut made which furnishes a long strip 5 millimeters square in cross-section, which is annealed, rolled and drawn.

Basse und Selve in Altena, Westphalia, use a crucible furnace of specially effective construction. A shaft of refractory material is surrounded by a jacket of double sheet iron, the space between the iron walls is filled with asbestos. Between this mantle and the fire clay shaft is a space into which air is pumped, with holes through the shaft to admit the air to the crucible space at four different levels and all around the crucible. The latter is packed in with coke, and stands on a pedestal of fire clay which has also air conduits to admit air around the base of the crucible. In this manner the fuel around the crucible is burnt very regularly and economically, it being found that only one-third as much fuel is used as in the old form of wind furnace; or, giving the actual figures, there were used for melting nickel, bronze or brass, per 100 pounds of metal melted, 85, 63 and 37 pounds of coke in the old-style wind furnaces, and 35, 19 and 11 pounds, respectively, in the new furnace. The latter is, moreover, portable, can be carried around the shop and the metal poured wherever needed, without removing the crucible.

The *Elmore Metall-Aktien Gesellschaft*, in *Schladern on the Sieg.*, make copper tubes electrolytically. The refined copper is granulated in water, and placed in a layer 16 inches deep on the bottom of a lead-lined tank, 19 feet long by 6 feet wide. The solution is copper sulphate with 3 per cent free sulphuric acid, and the horizontal rolls forming the cathode run in glass bearings at a distance of 1.2 inches above the

copper. The agate burnishing roller moves over the depositing copper once for every 0.03 to 0.04 millimeters of increasing thickness. The current density is 200 amperes per square meter of immersed cathode surface, or 800 amperes per tank: the voltage 1 to 1.25 volts per tank; at which 4.4 millimeters thickness of copper is deposited per week. It is highly important that there be no interruption during the deposition, as this would produce immediately an oxide layer, which would prevent complete adhesion of the subsequently deposited copper. This plant makes 1500 metric tons of copper tubes yearly; the largest tube so far made is 2.5 meters diameter by 6 meters long. The mandrels used as cathodes are of easily fusible brown alloy, the composition of which is kept secret; they are covered with Stanniol before being used. After deposition is finished, they are put into a superheated steam chamber and the alloy melted out. The deposited copper shows 36,000 pounds tensile strength per square inch, with 15 to 16 per cent elongation: after annealing, 30,000 pounds, with 50 per cent elongation. The slimes remaining in the tanks contain gold and silver, and are a source of profit, being sold for $450 per ton.

The English Electrometallurgical Co., Ltd., at *Leeds*, also use the Elmore process, with a capacity of 100 tons per day. The main precipitating room is 200 by 250 feet, and contains 216 precipitating baths. The mandrels here are polished copper tubes, well rubbed with grahite to prevent adherence of the deposited copper. The raw material is Chili blister copper containing 95 to 96 per cent of copper, and is cast at the works into slabs some 17 feet long, to fit the bottom of the tanks. Tests of the copper here deposited show from 50,000 pounds tensile strength per square inch, with 6 per cent elongation to 30,000 pounds, with 50 per cent elongation.

Julius and August Erbslöh, in Barmen, manufacture polished, silvered and gilded copper plate, rods and sheet of brass and tembac, and of aluminium. The aluminium slabs are from Switzerland or America, and are 3 to 4 inches thick. They are rolled down cold. After being annealed and then cleaned in caustic soda solution they are trimmed by hand, to free them from particles of slag, flaws, etc., and then rolled down further, to sheet, without warming, but being several times cleaned and annealed. The annealing furnaces are muffle furnaces with cast-iron muffles.

Smith's Dock Company, Ltd., South Shields, cast brass and bronze, and make a specialty of copper blast-furnace tuyeres. For producing sound castings of copper, the purest copper obtainable is used, since very small quantities of iron, arsenic or sulphur are very injurious. The copper is also preserved as carefully as possible from oxidation, while melting down, for when cuprous oxide is formed it dissolves in the metal and renders the production of sound castings impossible. This firm adds 1 to 1½ per cent of tin or zinc to the copper thus being cast; lead is not usually added, except in cases where the casting has to stand the corrosive action of acid water; phosphorus is used to the amount of 0.5 per cent for castings, with very thin walls. Lately, 1 to 5 per cent of manganese has been used, with best results in such castings, using 30 per cent manganese-copper; such castings are dense, hard, tough and easily worked. The castings under discussion are cast in dried sand molds.

[The preceding abstracts from Herr Westman's account of his "metallurgical journey" demonstrate the value of the Swedish custom of sending out experts in various lines to examine as far as possible and report as extensively as is permitted, on the metallurgical industries of other nations. It is a most admirable custom, both for the "traveler" and for the "stay-at-homes," and one which might be copied with great advantage by our American universities, societies and large industrial concerns.—*Abstractor.*]

Gold.

Cyanide Process.—The advantages of using filter presses are rapidly becoming recognized, and their use is being extended from the treatment of slimes, where they were first employed, to the treatment of fine sands; and it is even proposd by some enthusiasts in their use, to grind the whole of the ore to fine sand or slime and treat it all in the presses. C. Göpner, of Hamburg, writes interestingly on this subject in Borcher's *Metallurgie* for December 22.

According to Herr Göpner, he was the first to recognize the advantages of the filter press treatment, in 1892. The mines of Western Australia, just then opened, were having extraordinary difficulties with the amalgamation of their gold ore, the gold being so finely disseminated that in some cases, although containing over $100 worth of gold per ton, scarcely $10 per ton was extracted by amalgamating milling. The author obtained samples of this ore, had an experimental filter press built for him, and by agitating the very finely ground ore for two hours, with a 0.25 per cent cyanide solution, and filtering in the press, found that he extracted 96 per cent of the gold present, obtaining it in clear cyanide solution ready for precipitation.

The Hannans Brown Hill Gold Mining Co., of *Kalgoorly*, were the first to put the idea into practical use, with a plant of 100 tons daily capacity. The extraction obtained was 82.7 per cent of the gold from the coarse sands, 88.5 per cent from the average sands, and 93.1 per cent from the finely ground flour.

The presses used have 50 frames, each 40 millimeters thick and 1 meter square. The cubic content of these frames is 1.656 cubic meters, corresponding to about 3.5 tons of ore at one treatment. Much trouble was found in getting suitable filter cloth, until, after extensive tests, a cotton-duck cloth from the United States, such as is used for sail-cloth, was found to answer all requirements, and is now used exclusively. Using air pressures of 50 to 75 pounds per square inch, the press can be filled in 7 minutes; the washing, emptying, cleaning and reclosing take two hours and thirty minuthe; nine batches are treated per day, when the workmen become skillful.

The above work was all done by first agitating the slimes in tanks with cyanide solution, and then filtering in the presses. Experiments were made in running the ore into the presses with water, and then washing with cyanide solution in the press itself, but 10 per cent less gold was extracted in this manner, and the original practice was reverted to. For the maximum extraction by filter-press treatment, it is recommended that the ore be pulverized in roller-mills. The filter press is not, however, universally applicable, each case of ore and slimes must be studied for itself, and the best method of treatment for it discovered and applied.

At the present time, 80 per cent of all the ore mined in the Kalgoorly fiield is being treated in filter presses, amounting to 70,000 tons of ore and 15,000 tons of accumulated tailings monthly and yielding over $3,000,000 monthly: The ore is either ground directly to fines, agitated with cyanide solution containing cyanogen bromide, and filtered; or it is amalgamated, concentrated, the concentrates roasted and treated with cyanide solution. In any case, the slimes are treated altogether by filter presses.

Vanadium.

H. Herrenschmidt communicates to the *Comptes Rendus* of the French Academy an interesting paper on the extraction of vanadium (1904, Vol. 139, page 635), of which we abstract the following, from a translation in Borchers' *Metallurgie* (*December* 8). The process described is that used at the works of Bas-Coudray, near Le Genest (Mayenne). The ore is lead vanadate, from the Santa Marta Mine in Spain, and contains 12 to 14 per cent of vanadic oxide to about 50 per cent of lead. It is melted in a reverberatory furnace with carbonate of soda and coal, forming metallic lead, which collects also the silver in the ore, and a slag containing sodium vanadate, sodium aluminate and iron silicate. The slag is kept melted in another furnace, and air blown over it until the

vanadium oxide is fully oxidized. The mass is then run into boiling water, granulated and leached. After repeated washings, the residuum does not contain over 2 per cent of vanadic acid; 95 per cent of the vanadic acid in the ore is thus obtained in solution. Silica is removed from the solution by careful addition of sulphuric acid, and filtering. The vanadic acid is then precipitated by adding excess of sulphuric acid, evaporating and washing. The precipitate contains some 5 to 8 per cent of impurities.

If ferro-vanadium is the object, the solution of sodium vanadate can be precipitated by adding iron sulphate, and sodium carbonate, in such proportion that two parts of iron are precipitated for every 1 part of vanadium. The precipitate is filtered out, dried, mixed with aluminium powder and reduced to an alloy of 33 per cent vanadium to 66 per cent iron, by Goldschmidt's process. If nickel sulphate, copper sulphate or cobalt sulphate are used, instead of iron sulphate, alloys of vanadium with nickel, copper or cobalt are obtainable in a similar manner. Vanadium-nickel alloy is also made by mixing vanadic acid with nickel oxide, adding carbon, mixing, pressing into cubes, and strongly heating in a carbon-lined crucible. The alloy thus obtained, however, is not so homogeneous or free from impurities as that made by reduction by aluminium, but the process is cheaper.

BOOK REVIEWS.

GRUNDZÜGE DER SIDEROLOGIE: III. Die Hüttenmännischen Prozesse. By Hanns Freiherr von Jüptner, Leipzig: Arthur Felix, 1904. 422 pages. Price, marks 18.00.

This third part completes von Jüptner's treatise. It has appeared in two sections, the first treating the behavior of iron towards various reagents or other elements, the second of the metallurgical processes for producing iron and steel.

These treatises have very marked characteristics, making them take a place in the metallurgy of iron never before so satisfactorily filled. They are, in brief, treatises on the chemistry and physics of iron, presenting the scientific principles on which its metallurgy is based. For descriptions of processes, mechanical details and illustrations, the reader must look elsewhere, such as to the works of Turner, Sexton, Wedding or Yedebur; for elementary explanations for a beginner, Bauerman's many-editioned work would be recommended in preference; but for the experimental results by which a thorough insight is given the metallurgist into the scientific principles involved in the study of iron, von Jüptner's work is incomparable.

The subject is so vast, and its scope so various, that no one man could be an expert on the whole of it. Von Jüptner's work has therefore been mostly that of a compiler, and, as such, there will be found minor shortcomings, such as a repetition of results at places without as intelligent a criticism of their value as might have been given. But, the work will be of most use to that class of metallurgists who simply want the data, and are often as competent as the author himself to make the evident criticisms, and to properly weigh the value of the information. While not a work of genius, yet it has been patiently, carefully and meritoriously done, and the author has gained by it the well-deserved thanks of the metallurgical world.

The Brass World.—The first number of a new journal, under the title *The Brass World and Platers' Guide*, has just been issued. The journal will be devoted to the art of refining, alloying, casting, rolling, founding, and electroplating of all the non-ferrous metals. Mr. Erwin S. Sperry, of Bridgeport, Conn., is the editor. Some contributions to the first issue refer to fire cracks, their cause and prevention; black nickel-plate and its production; a new cheap white metal for sand-casting, and numerous smaller notes. We wish every success to our new contemporary.

Producer Gas Engine Electric Plants in Chemical and Metallurgical Works.

BY FRANK C. PERKINS.

Electric power is now being extensively used, not only in the largest chemical works but also in all of the recently erected iron and steel plants. The high power gas engine is being introduced in the largest plants in England, Germany, Austria and other Continental countries, directly for power purposes as well as for generating electrical energy for lighting, power and chemical processes. Thus, two 250-hp. Deutz double-acting, single-cylinder engines, direct connected to continuous current electric generators, have recently been installed at the chemical works at Kalk, near Cologne, Germany.

In large chemical works gas producers are frequently employed for supplying the fuel for high power gas engines in the electric power stations, the economy of the entire installation being equal to, or better, it is claimed by prominent engineers, than the best steam engine power house equipment. Fig. 1 shows a gas producer plant of 10,000-hp. capacity erected at the Castner-Kellner Alkali Co.'s chemical works at Runcorn. This installation was furnished by the Power-Gas Corporation, Ltd., of London, and is used for furnace

FIG. 1.—GAS PRODUCER PLANT.

work as well as for supply gas for a number of Koerting gas engines, each having a capacity of 700 B. H. P.

Another large gas producer plant has recently been installed by the same company at the steel works of William Beardmore & Co., of Glasgow, Scotland. The ultimate capacity of this installation will be 200 tons per day, but at the present only five producers are in operation. The gas is used for the steel furnaces as well as to drive Koerting and Oechelhauser gas engines of 1000 and 500 B. H. P. each.

Gas producers are necessary at iron and steel works for supplying fuel to gas engines when the waste blast furnace gases are not adequate for providing sufficient power, but at present high power gas engines utilize these waste gases extensively, and with the greatest of economy. Fig. 2 shows a 300-hp. single-cylinder, double-acting Deutz gas engine driving a direct-current generator at the power plant of the Lauchhammer Iron Works.

The suction gas producer is being largely employed; the advantages claimed for it are that no boiler nor gas holder is necessary, and that it can be erected without danger in

any convenient place, while it is easy to start and operate. With the suction gas producer plant only a small floor space is required, and there is no soot, smoke or odor, and the highest and best utilization of the fuel is obtained, there being automatic generation of the gas by the engine. Combinations of suction gas producers and gas engines are built, for

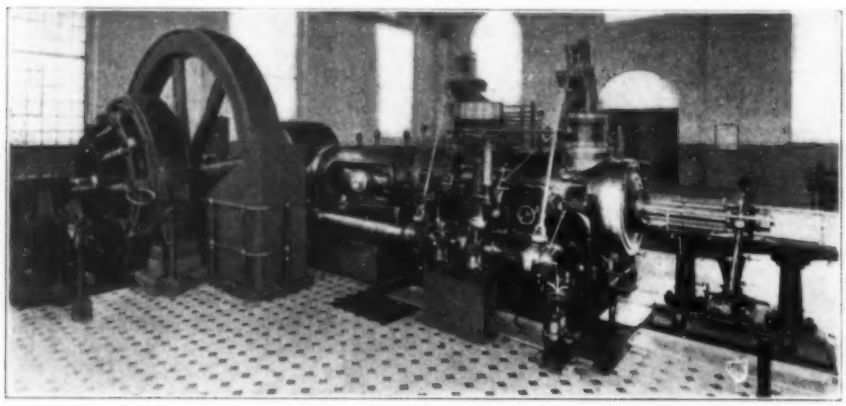


FIG. 2.—BLAST FURNACE GAS ENGINE ELECTRIC PLANT.

instance, by the Gas-Motoren Fabrik Deutz, represented in this country by the Otto Gas Engine Works, of Philadelphia.

It is claimed by prominent engineers that a high advantageous utilization of fuel results from employing gas engines which take their supply from producer gas power plants constructed for generating out of a suitable fuel a proper mixture of hydrogen and carbon monoxide. Some of the gas producer plants of the pressure type have boilers for providing steam which is mixed with air; by means of a fan or steam jet blower it is forced through the glowing fuel, with the resulting oxidation of the fuel to carbon monoxide, while hydrogen is simultaneously obtained from the steam. The gas engine receives these combustible gases from a gas holder, to which they are conducted after passing through scrubbers.

The gas producers of the suction type are said to be cheaper and simpler, requiring no gas holder or boiler. In these plants an evaporator takes the place of the boiler, being con-

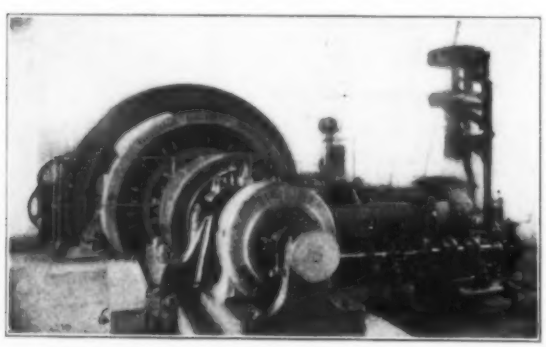


FIG. 3.—COKE OVEN GAS ENGINE ELECTRIC PLANT.

structed on top of the producer in some cases and in the form of a tubular evaporator in larger installations. The steam is generated in the evaporator by taking advantage of the heat of the producer and of the gases, the firing of an independent boiler being avoided. It is claimed by engineers favoring this type of producer that it utilizes 80 per cent of the fuel and the grate is kept cool by passing the steam under the same. It is also claimed that the suction gas producer can start quickly from a part load to an excessive load, as the carbon monoxide is formed and the steam provides the hydrogen very rapidly. The gas producer is said to work well with either coke, charcoal, or anthracite coal, supplying the gas engines with an abundance of fuel at high economy.

Where waste gases are obtainable from either coke ovens or blast furnaces, the gas engine is particularly desirable and is largely employed in Europe, and is now being introduced in this country. Fig. 3 shows a twin-cylinder, single-acting gas engine of the Cockerill type operating in Witkowitz. This coke oven gas motor operates a 150-kw. three-phase generator by direct connection at a speed of 160 r. p. m. The gas engine was constructed by Breitfeld, Danek & Co., of Prag-Karolinenthel, Austria, Hungary, and the three-phase alternator by Ganz & Co., of Budapest-Leubersdorfs, the frequency being 50 periods per second, and the pressure 575 volts.

The earlier electric stations operated by gas engines supplied with producer gas were not of the direct-connected type, but were belted to the dynamos as at the gas engine electric power plant at Uster, near Zurich, Switzerland. A number of gas producer systems are now in operation successfully in Europe and America, and there is every reason to believe that during the next decade the gas engine will occupy an important place in the electric light and power stations throughout the world as the high efficiency of the entire installation compares well with the best plants using other prime movers.

Revolving-Hearth Roasting Furnace.

The development in late years of so many mining properties containing large quantities of comparatively low grade and refractory ores, has created a demand for a furnace of large capacity, in which such ores can be roasted preparatory to chemical or further igneous treatment.

Heap and stall roasting and the hand reverberatory roasting furnaces have, in turn, made way for furnaces in which the ore is stirred or rabbled by mechanical means. The designs of the first mechanical roasting furnaces followed closely along the lines of the hand reverberatories, but the later designs have been, in nearly every instance, furnaces having a round hearth and revolving rabbles. In the Holthoff Revolving-Hearth Furnaces, the design of Mr. Henry C. Holthoff, of Milwaukee, Wis., this principle is reversed. The hearth, which is annular in form, is made to revolve about a centrally located gas producer. The wall of this producer forms the inner wall of the roasting chamber and support for its arched roof, while the outer wall is supported on cast-iron columns or structural steel.

The gas generated in the producer enters the roasting chamber through a number of pipe channels radially disposed in the walls of the producer. Air pipes are also located in the walls of the producer, the lower ends opening to the atmosphere through a regulating valve, the upper ends extending into the roasting chamber in close proximity to the gas exits. The air, heated in its passage through the producer walls, mixes with the gas upon extrance to the roasting chamber, and combustion is spontaneous. The flame traveling radially across the hearth finally discharges through a series of tile pipes into a large circular smoke flue supported on steel frame work just above and around the outer edge of the furnace. It will be noted that while the hearth resolves, the main furnace structure is stationary.

The rabble arms, of which there are four sets, extend down through the arched roof of the roasting chamber, and all cooling water connections and adjustment of the angularity of the rabble blades are accomplished outside of the furnace. The rabble arms consist each of two pieces of cast-iron pipe of different diameters, one within the other, the outer pipe being tapped into the hollow rabble blade. This arrangement is a simple and effective means for water-cooling the rabbles.

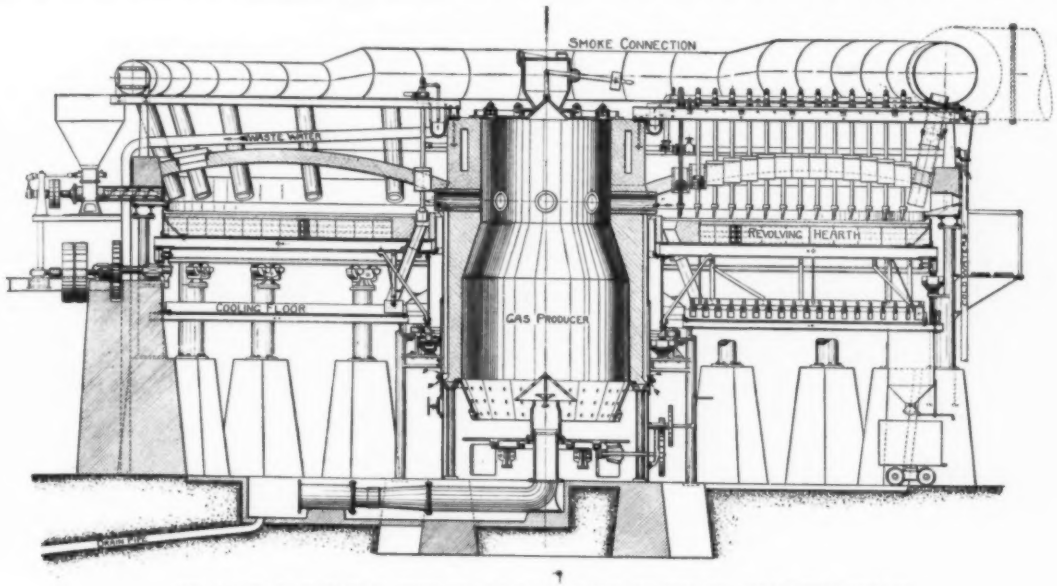


FIG. 1.—CROSS-SECTIONAL VIEW OF HOLTHOFF REVOLVING HEARTH ROASTING FURNACE.

The ore, fed through a feeder having an adjustable eccentric and a spiral conveyor extending through the outer wall of the furnace, drops on to the outer edge of the hearth, and, as the hearth revolves, is rabbled across same in a spiral path. Reaching the inner curb of the revolving hearth, it drops through discharge spouts on to the inner edge of the stationary cooling floor located directly under the revolving hearth. It is then moved across the cooling hearth and finally discharges at the outer edge of the cooling floor into a revolving hopper or to a belt conveyor or elevator.

The hearth revolves upon equalizing roller bearings of conical form, and is revolved by means of a roller rack and a pinion. From the inside of the mantel upon which the outer wall rests, a steel metal apron is hung, extending down inside the outer curb of the hearth, and the top of the ore bed being higher than the bottom of this apron, the opening between the hearth and the wall of the roasting chamber is effectually sealed against the entrance of cold air. By means of valves or dampers located in the air inlet and smoke discharge pipes, the temperature in the roasting chamber is under complete control of the operator at all times, and, as the angularity of the rabble blades can be changed quickly and easily, the time the ore remains in the furnace is also under complete control. The angularity of all the rabbles of one set is changed by the movement of a lever conveniently located for the operator, and which slides in a quadrant on which are figures indicating the degree of angularity of the rabbles.

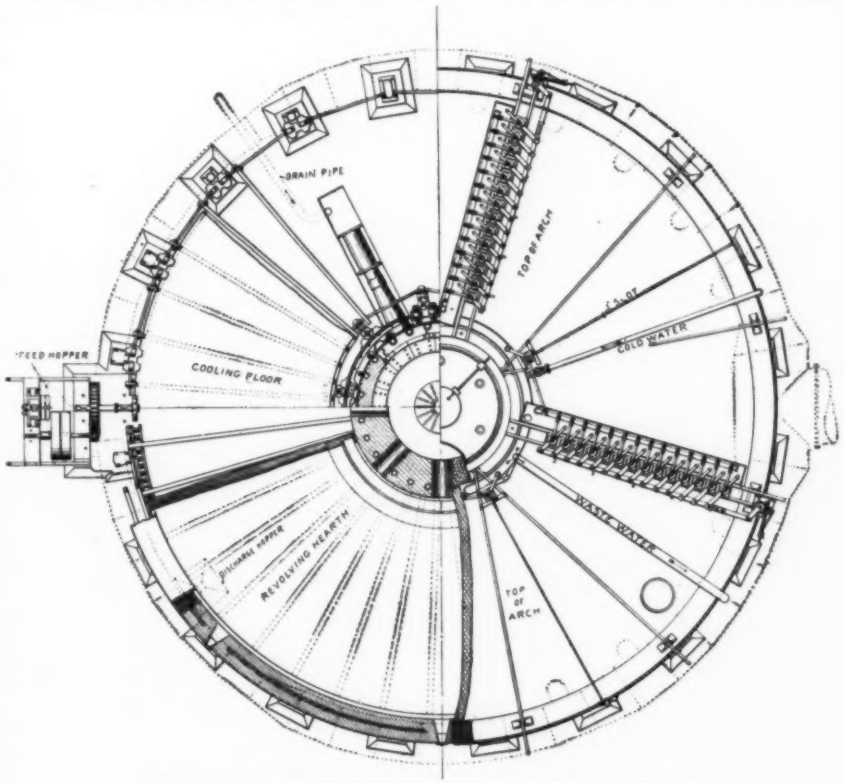


FIG. 2.—PLAN OF HOLTHOFF REVOLVING HEARTH ROASTING FURNACE.

The furnace above described was designed especially for

treatment of low sulphur ores, such as are found in the Cripple Creek district and was built about a year ago for the Portland Gold Mining Company's mill at Colorado City.

In a two months' test run of this furnace the following results were obtained, the figures given showing the average per day of twenty-four hours during the entire test.

Tons of raw ore charged	99.82
Per cent Sulphur in raw ore	2.572
Per cent Sulphur in roasted ore (sulphides)	.142
Tons of coal consumed	9.45
Pounds of coal per ton of ore charged	189.80

These furnaces are also built in three other types, one similar to that described above, except that a plain fire-box is substituted for the gas producer. Another type designed for roasting sulphide ores preparatory to smelting, is built along the same general lines as that described above, except that the cooling floor is omitted and the roasted ore discharges automatically into a stationary receiving hopper from which the calcines are drawn off into cars or buggies and taken to the smelting furnace.

The fourth type of furnace is designed for roasting high sulphur pyritic ores. In this type of Holthoff furnace the cooling floor is omitted and a plain fire-box is substituted for the gas producer. This fire-box, however, is used only to ignite the sulphur in the charge. After ignition, the sulphur burns spontaneously, and the fire-box is used merely as an air flue.

The advantage claimed by the builders of these furnaces, the Power & Mining Machinery Co., at Cudahy, Wis., are: First, great capacity. Second, ability to control the temperature in the roasting chamber, as well as the rate at which the ore is moved across the hearth, thus giving the operator complete control of the roast during the entire operation. Third, by using a gas producer incorporated in the furnace structure, no heat is wasted in piping, and the expense of installation and repairs for piping, found in furnaces in which the fire-box is separate from the furnace proper, is eliminated. Also, by generating gas in the producer, the combustion of which takes place in a roasting chamber, the greatest possible utility is made of the heating value of the fuel, and it will be noted from the figures given above that the fuel consumption per ton of ore charged is exceedingly low. Fourth, the small amount of manual labor connected with the use of these furnaces is an important item. The only labor required is for lubricating and such general attention as is due any piece of machinery however simple, and one man can attend several of these furnaces. Fifth, throughout the general construction of these furnaces, provision is made for expansion and contraction of the various parts, so that the users are saved the annoyance and expense of repairs from this cause, which are so frequent in other furnaces.

The Power & Mining Machinery Co., Cudahy, Wis., are issuing their Bulletin C-1, describing this furnace, which, we are informed, will be sent gratis on request. This bulletin contains complete detailed description of the design, construction and operation of these furnaces.

Personal.

Mr. Ernest Kilburn Scott, of London, England, has been appointed to organize and take charge of the Electrical Engineering Department at Sydney University. Mr. Kilburn Scott's name is well known to our readers as a virile technical writer of very wide and detailed knowledge. He recently contributed to our columns some information regarding the manufacture of a refractory furnace lining material from magnesite. Mr. Scott carried out this work for Mr. Turner, the owner of the magnesite mines at Salem, in Southern India, who has recently been awarded a gold medal at the St. Louis Exposition for such furnace linings.

Prof. Walter Nernst, of the University of Goettingen, the distinguished founder of the osmotic theory of galvanic cells and inventor of the Nernst light, has accepted a call to a chair for physical chemistry at the University of Berlin.

While going to press, we learn with deep regret of the sudden death, on January 25, of Mr. E. H. Mullin, of the New York office of the General Electric Co. Mr. Mullin had been ill with grip for four or five days.

Digest of U. S. Patents.

Prior to July, 1902.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Molten Electrolytes, Miscellaneous.

(*Concluded from page 48.*)

679,253, July 23, 1901, A. H. Cowles, Cleveland, Ohio.

Volatile elements such as sodium, potassium, zinc and phosphorus, are extracted from fused compounds by electrolysis, the specific example given being the extraction of sodium from a bath containing sodium fluoride and aluminium fluoride in which sodium carbonate is dissolved. Upon electrolysis oxygen is set free at carbon anodes, while the sodium is deposited upon a porous hearth produced by slowly baking a mixture of coke, granular carbon, and a heavy hydrocarbon. The bath is maintained above the volatilizing temperature of sodium, under which conditions the sodium passes through the porous hearth and is condensed in the surrounding casing. The condenser is preferably heated in order to permit the removal of the sodium by tapping. The vessel is of metal, the sides being cooled, and protected by a layer of the solidified electrolyte. A receptacle employing twenty carbon anodes, each 2½ inches in diameter, will carry about 3000 amperes at 5 volts. The bath should be periodically renewed by addition of sodium carbonate.

679,997, August 6, 1901, George P. Scholl, Philadelphia, Pa.

For the electrolytic separation of sodium employs a bath containing caustic soda and sodium sulphide in equal parts. The bath has a somewhat higher melting point than sodium hydrate, but has the advantages that the e. m. f. required for decomposition is lower, and that less hydrogen at a lower temperature is liberated at the cathode. It results from the latter advantage that a single cell may be safely enlarged and greater economy of operation secured. The electromotive force is regulated to decompose the sulphide only, the liberated sulphur reacting with the hydrate to again form sulphide. Additional quantities of hydrate are supplied during the operation. The method is stated to be applicable to metals other than sodium, the required condition being that the electrolyte comprise a sulphur compound and a fusible compound readily converted into a sulphide.

699,851, May 13, 1902, C. W. Roepper and G. P. Scholl, Philadelphia, Pa.

An apparatus for the electrolysis of fused salts, designed particularly with a view to use in large installations. The chief features are the provision of separate intercommunicating heating and electrolytic chambers; the employment of a number of electrolytic chambers grouped around or otherwise arranged in conjunction with a single heating chamber; a system of jacketing whereby a thin incrustation of the solid electrolyte protects parts from the corrosive action of the bath; and a construction which provides for the ready replacement of parts which are subjected to the most rapid deterioration. Many alternative constructions embodying the foregoing features are shown. The fusion of the electrolyte is accomplished by gaseous products of combustion passing over the material; the latter may be protected from the gases by a floating layer of carbon, sand, or melted slag. No specific example of the use of the apparatus is described.